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CANADA
DEPARTMENT OF MINES
HON. CHARLES STEWART, Minister; - - CHARLES CAMSELL, Deputy Minister

MINES BRANCH
JOHN McLEISH, Director

SUMMARY REPORT

OF INVESTIGATIONS MADE BY THE

MINES BRANCH

DURING

THE CALENDAR YEAR ENDING DECEMBER 31

1921

OTTAWA
F. A. ACLAND
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SUMMARY REPORT OF MINES BRANCH INVESTIGATIONS, 1921.

MINERAL RESOURCES DIVISION

I

COALMONT AMBER

Alfred W. G. Wilson

It has been known for some time that fossil resin occurs in the coal seam at Coalmont, B.C., and it has been stated recently that the quantity present is sufficient to warrant commercial recovery. A sample of glossy black coal containing about 10 cubic inches of this resin was submitted to this office for examination and identification in December of 1921.

The resinous material possesses characteristic lustre and fractures conchoidally. Many of the fragments are translucent and pale amber in colour. The main portion of the sample is almost opaque, but in reflected sunlight fracture surfaces are strongly fluorescent and of a greenish tone. Other samples are of a cherry colour and very clear.

Fossil resins of this kind have previously been reported in small amounts from various Cretaceous coals in western Canada. A natural beach deposit of similar material also occurred at Cedar lake, Saskatchewan. This deposit was recently staked as a mineral claim, but close examination showed that most of the commercial amber which it contained had been removed and marketed in previous years. The Cedar lake deposit was originally discovered and reported by J. B. Tyrrell, in 1890, and the samples of fossil resin collected by him were examined by Dr. B. J. Harrington at McGill University in 1891. (Dr. Harrington's description of the Cedar lake fossil resin is published in the American Journal of Science, Vol. XLII, October, 1891, pp. 332-335. Extracts from this article are published in Volume V, Part I, Report E, Geol. Surv. Can., 1890-91, pp. 225-228.)

Fragments of the fossil resin from Coalmont were submitted to Mr. R. T. Elworthy for chemical examination. They yielded 60 per cent of volatile products, between the temperatures of 300° C. and 410° C. There was only a very small yield of succinic acid, on which basis the material would be classified mineralogically as retinite. No attempt was made to fractionate the amber oil obtained. Nearly half the residue after distillation was soluble in turpentine. The insoluble balance consisted largely of carbon, some of which may have been present as an impurity derived from the coal, and some of which was undoubtedly due to partial decomposition of the sample tested. If distillation had been stopped at a lower temperature there would have been less decomposition, less amber oil would have been recovered, and a larger proportion of turpentine-soluble colophony would have remained.

The specific gravity of different fragments varied between 1·03 and 1·10. The coal has a specific gravity of 1·31. It was found possible to separate the two easily by using solutions of either sodium chloride or calcium chloride diluted to a specific gravity of about 1·20.

The material is slightly harder than the higher grades of Baltic amber used in the manufacture of beads, pipe stems, and ornaments. The finer hand picked pieces, if available in sufficient quantity, would be suitable for use in this way if free from flaws. Smaller pieces, provided they are free from impurities, could be pressed into sheets and blocks suitable for commercial purposes. The bulk of the material will probably be found to be suitable for the manufacture of varnishes or lacquers for use in electrical work.

The immediate problem before the operators of this mine appears to be the question of separating the commercial amber from the coal. Arrangements have been made to conduct a series of experiments on the concentration of this product at the testing laboratories of the Mines Branch.

II

THE IRON INDUSTRY OF BRITISH COLUMBIA AND ONTARIO

A. H. A. Robinson

British Columbia

In British Columbia, little progress has been made towards the establishment of the much mooted iron smelting industry. So far it has not been possible to get capable and experienced iron masters financially interested in such a project. No insuperable technical difficulties appear to be anticipated, but the commercial side of the problem, especially as regards an assured market for an output of pig iron large enough to make operations profitable, is somewhat doubtful.

As regards electric smelting, at the time of my visit one firm in Vancouver was making steel from scrap in the electric furnace, very profitably it is said, but the prices quoted for electric power would appear to put out of consideration, for the present at least, any question of electric smelting for pig iron as a commercial proposition.

Ontario

Early in 1921, a company known as the Palatinate Mining Corporation, said to be a subsidiary of a large Polish American co-operative society having its headquarters in Chicago, secured an option on the Atikokan Iron Company's blast furnace at Port Arthur, with the declared intention of putting it into blast on ore from the old Paulson (magnetite) mine, just south of the international boundary in Minnesota. They also secured from the city of Port Arthur an option on 200 acres of land as a site for steel works. A start was made at overhauling the blast furnace plant and a little work was done in extending the Port Arthur, Duluth and Western

railway (C.N.R.) from North lake to Paulson mine. A change appears then to have been made in the original plans and the idea of using the Paulson mine ore abandoned. At the time of my visit, in October, work was practically at a standstill, and the Company was said to be considering the acquisition of the Atikokan magnetite mine and the Loon lake hematite deposits, both of which are in Ontario. Up to date no active operations have been reported.

During 1919 and 1920, seventeen diamond drill holes, having an aggregate length of 7,274 feet, were put down in one of the iron ranges east of lake Nipigon, by Cleveland, Ohio, parties. They proved up considerable quantities of low grade material, but no workable body of ore of present day merchantable grade was encountered.

In the Soo district, the Algoma Steel Corporation's Magpie mine was closed early in March, and for the balance of the year the supply of Magpie ore required by the furnaces at Sault Ste. Marie was drawn from stock piles at the mine. There has been no iron ore mined in Canada since the closing of the Magpie.

In July, 1921, bush fires destroyed completely the buildings and equipment at the Algoma Steel Corporation's Helen mine—both that remaining at the old Helen plant and that more recently installed to develop the siderite ore-body.

During the year some diamond drilling was done north of the Soo, on the old Hillier magnetite claims, which lie just south of the upper Goulais river and about 9 miles east of the Algoma Central railway, in ranges XII and XIII, township 22. The work was being done by Messrs. Smith and Travers, diamond drill contractors, for a Sault Ste. Marie syndicate. The Algoma Steel Corporation, in anticipation of possible developments in the utilization of low grade magnetites, restaked a number of mining claims in the same locality.

Experimental work was resumed at Moose Mountain during the year, under the supervision of Mr. A. J. Anderson.

III

IRON OXIDE PIGMENTS IN ONTARIO

Howells Fréchette

In continuation of the investigation of mineral pigments in eastern Canada, many reported occurrences of ferruginous pigment were visited and examined in the southern part of Ontario.

In the section of the province visited there are apparently no large deposits of high grade iron oxide pigments such as have been so extensively worked near Three Rivers, Quebec. While several deposits have been worked in the past, the more exacting requirements of the paint trade, and the necessity for producing in large quantity in order to meet competition, have forced discontinuation in all cases. It would seem that the greatest chances for economical production are with certain of the clays and shales, especially in Halton county, which can be used for cheap paints and as a coloured filler for linoleum.

LEEDS COUNTY

In Leeds county there are a number of small deposits of iron ochre in swampy ground, in most cases adjacent to Potsdam sandstone or Beekmantown dolomite. The deposits probably owe their origin to iron leeched from these rocks. Such deposits were seen on lot 20, concession II, and lot 21, concession VII, Bastard township, and lot 24, concession X, Leeds township. None of these can be considered of commercial interest.

On lot 18 or 19, Broken Front, Yonge township, there are two deposits which, about forty years ago, were exploited for raw material for paint making. One is composed of earthy hydrated oxide, which has been deposited in a swamp by a spring issuing from rocks carrying much iron pyrites. The greater part of the deposit has been removed and what remains is very much contaminated with sand. When the uncalcined material is ground in oil it produces a fairly dark sienna paint with a slightly green tone. When calcined it produces a red paint lacking in intensity.

The other deposit is of a hard stony character, and the outcrop is so weathered and broken that a proper idea of the type of occurrence could not be formed. When ground in oil without previous calcining the paint produced is a rather light coffee brown, and the calcined material produces a bright reddish brown.

In this deposit there are many small stringers and inclusions of quartz which would interfere with the grinding of the material for paint making, but much of the quartz could probably be removed by screening during the grinding operations.

RENFREW COUNTY

While no deposits of value are known in this county, small deposits were seen on the north side of the old road which parallels the railway on lots 9 and 10, concession II, Blithfield township, in small catchment basins in a drainage course leading to a large swamp.

Since the rocks in the vicinity carry much pyrite, prospecting along the margin of this swamp might disclose deposits of bog iron ore, or ochre.

At the foot of a gully leading northwest from the Mile Lake pyrite mine on lot 1, concession I, Blithfield township, a deposit of sandy ochre was seen on the edge of a large swampy flat. This also would be a place worth prospecting.

HASTINGS COUNTY

From time to time occurrences of iron ochre have been reported from the neighbourhood of Madoc. A number of lots were visited in Huntingdon and Madoc, and many enquiries made in the vicinity, but no deposits of any interest were seen.

Several deposits of iron ochre are known to occur in the northern part of Hastings county. These will be visited in 1922.

MUSKOKA DISTRICT

In the townships of Oakley and Draper there are deposits of bog iron ore on many lots. These were visited in the hope of finding ochre associated with the hard bog ore. A few small patches of clean ochre were seen, but none of them could be worked for pigment.

SIMCOE COUNTY

About two thousand feet from the shore of Georgian bay, on lot F, concession XIX, Tiny township, springs issue from the face of a hill below an extensive sandy flat. These springs deposit iron ochre and have formed a number of small beds, with a maximum depth of about two feet. The ochre is much intermixed with sticks and leaves and is somewhat sandy. While paint of good colour can be made from the ochre, these deposits cannot be regarded as of any commercial value.

On lot 1, concession XI, Nottawasaga township, just below the falls of the Noisy river, springs issuing from the rocks of the river bank deposit calcareous sinter and iron oxide. In some places the amount of iron oxide is sufficient to strongly colour the material. Samples tested produced paint of a rather light umber colour, but the calcined material gave a very poor shade of brown. The deposits at one time, it is said, covered several hundred square yards to a depth of about four feet. Slides and floods have reduced the area to a few square yards. Owing to the smallness of the deposit, and the calcareous nature of the material, it is of no value.

GREY COUNTY

In the 1863 Report of the Geological Survey there is a record of a deposit of yellow ochre at the base of the hill "on the southwest side of the village of Owen Sound." The deposit was said to be four feet thick in places. This deposit could not be found as it has probably been built over.

Small patches of ochre are to be seen covering the rocks at various points along the face of the escarpment which surrounds the city. None of any value were seen.

BRUCE COUNTY

There are traces of ochre in many parts of Brant township, but no workable deposits were observed, although an effort was made to locate a deposit of ochre which is said to have been worked at one time.

HALTON COUNTY

Paint has been manufactured from Medina shales or from clay resulting from the weathering of these shales at several places in Halton county, but none has been produced in the last twenty years.

The Ontario Mineral Paint Works, Campbellville, procured clays and shale from lot 11, concession IV, Nelson township. This they ground and sold dry to paint manufacturers. Six colours or shades are reported to have been produced, namely red, drab, slate, olive and brown.

The above mentioned lot was visited and the deposits examined. The red clay and shale predominate. The shale is free from grit and grinds readily. In the uncalcined condition it produces a chocolate brown, and after calcining a brownish red. In places the clay is very sandy, while elsewhere it is comparatively free from sand. It produces paint of similar colour and quality. In both cases the paint has good opacity but poor tinting power.

The clays of other colour are only in small amount and are of little importance.

Similar clays and shales were prepared as pigment many years ago at Limehouse.

A sample of the red clay was taken on lot 15, concession IV, Esquesing, where there is a good development of it. It proved to be very free from grit.

The principal uses of these clays as pigment would be in the manufacture of cheap paints, or as fillers for linoleum.

NORFOLK COUNTY

Very extensive sandfields occur in the southern part of Norfolk county. Iron oxide, derived from these sands, has been deposited at many points as ochre or bog iron ore. The ochre for the most part is sandy and not in workable quantity, although one deposit was worked many years ago, on lots 12 and 13, concession IV, Walsingham township.

Deposits were visited on lot 12, concession VII, Charlotteville township; lot 15, concession I north of Talbot road, Middleton township; lot 12, concession XIV, Windham township; and lots 12 and 13, concession IV, Walsingham township.

On the two last mentioned lots a deposit of sandy ochre five or six acres in extent and about 1½ feet thick was worked at various times, the last operations having been conducted by the Ontario Mineral Paint Co., about twenty years ago.

The best parts of the deposit have been removed, and what remains cannot be considered as of commercial grade.

Samples taken in this county produced paints of good colour, opacity and tinting power, but were mostly characterized by their grittiness. When ground, uncalcined, in oil the colours produced ranged from Peruvian ochre to raw sienna, and the calcined material produced good and rather brilliant oxide-red paint.

MIDDLESEX COUNTY

Old departmental reports refer to deposits of ochre in Middlesex county, but even after close enquiry in several sections of the county no deposits of more than a couple of hundred square feet could be located. They were of no value.

IV

INVESTIGATION OF MISCELLANEOUS NON-METALLIC MINERALS

H. S. Spence

A report on the barium and strontium minerals of Canada, entitled "Barium and Strontium in Canada," was completed during the year, and will be published during 1922.

Field work consisted in the examination of a number of deposits of talc and soapstone, as well as of occurrences of fluorite and feldspar, all the localities visited being in Ontario and Quebec.

The data on talc and soapstone will be incorporated in the writer's report on these materials, which is in course of preparation, and which will probably go to press during 1922.

Ontario

The occurrence of so-called soapstone or potstone in the Lake of the Woods region has been mentioned in earlier reports of the Geological Survey.

Lake of the Woods.—Some interest was evidenced a few years ago in a deposit of soapstone at Pipestone portage, between Moore and Andrew bays, Lake of the Woods, 12 miles south of Kenora.

The writer visited this locality in company with E. F. Kendall of Kenora, one of the persons interested in the deposit.

About 100 tons of material from this deposit is stated to have been taken out some years ago, and shipped to the Dryden Timber and Power Company (now the Dryden Pulp and Paper Company), of Dryden, Ont., who had it sawn into blocks for lining the smelting furnaces of their sulphate paper plant. The material is reported to have proved too seamy for the above purpose, and to have crumbled badly.

The stone was obtained by blasting out the side of the low ridge which divides the two bays, and was loaded directly into a scow, and towed up the lake to Kenora station, on the main line of the Canadian Pacific railway.

The deposit consists of a band about 75 feet wide, striking approximately east and west, and running diagonally across the neck of land separating the bays. The dip of the band and of the enclosing slates is nearly vertical. The band is well exposed in the face of the 20-foot bluff at the northwest end of the 100-foot portage on Moore bay.

The material of this band, while described as a soapstone or potstone, is more in the nature of a soft chloritic slate of a dark, greyish green colour. The stone is rather irregularly jointed, and possesses a somewhat schistose structure, but is massive enough to break out in large blocks, and stands up well under the hammer. The stone is rather harsh to the feel, but is soft enough to be cut with an ordinary saw; the powder possesses little or no slip. The main constituents of the rock appear to be chlorite and dark-coloured ferro-magnesian minerals. Little talc is apparent. The material cannot be classed as a true soapstone, and, apart from its softness, bears little resemblance to soapstone.

Wabigoon lake.—An occurrence of soapstone was examined one mile west of Wabigoon station, on the main line of the Canadian Pacific railway, about 95 miles east of Kenora. The mining location is HW133, township of Zealand, and the owner of the property is E. G. Pidgeon, Wabigoon, Ont. The soapstone outcrop lies 500 yards south of the railway track.

Beyond a small amount of stripping, no work had been performed on this deposit prior to the writer's visit. The body of soapstone is exposed on the top, and on the northwest slope of a low knoll, and is well situated for working. There is very little soil overburden.

From an examination of the outcrops, there appear to be two parallel bands of soapstone, lying between granite walls, and striking northwest. The band on the north side has been proved for a length of 500 feet, with a width of 35 feet. This band is separated from that to the south by 75 feet of hard igneous rock, having the appearance of a greenish granite.

The south body is probably the larger of the two, but its extent had not been determined at the time of the writer's visit.

The north band, to which most attention has been paid, consists of a massive, soft, and compact, dark green stone, that can be easily cut by an ordinary saw. The stone possesses no schistosity, and is of fairly uniform, medium grain. Only surface material was available for examination, but tests made show it to possess good strength, in spite of the rather flawed nature of the test pieces. It is hoped to secure more representative material during 1922, upon which a complete series of tests may be conducted. The outer two feet of this band, along the contact with the wall-rock, consists of rather hard grit (siliceous) stone, but between this zone and the main portion of the body, there occurs a narrow, 18-inch band of very soft and fine-grained soapstone.

The prospect appears to be a promising one, from surface indications, but it will be necessary to await further development work before the true nature of the stone can be determined. The surface material can be readily cut into blocks that possess high crushing strength, and retain heat well. It is hoped that the stone may prove to be suitable for cutting into bricks for lining the smelting furnaces of sulphate pulp mills, as well as for general electrical and heat insulation purposes, switchboards, laboratory slabs, etc. The whole of the stone used for such purposes in Canada is at present imported from the United States.

Although the term soapstone has been used to designate the stone of this deposit, the material is probably not a true soapstone. It resembles closely the Alberene stone of Virginia, which is used extensively for the purposes mentioned above.

Lanark county, township of Lavant, concession III, lot 24 E. $\frac{1}{2}$.—A body of talcose dolomitic serpentine occurs on this lot. It is well exposed in a low ridge, and the outcrop extends for 300 feet, with a width of 50 feet. A few small pits have been opened at the base of the ridge and disclose a white dolomitic serpentine, carrying foliated and fibrous talc. Quartz is also present as small irregular inclusions.

The deposit lies one-fourth of a mile from the Kingston and Pembroke branch of the Canadian Pacific railway, and 2 miles north of Flower station.

While analysis of this rock has not yet been completed, its behaviour with acid would indicate that it contains considerable carbonate, probably dolomite.

Quebec

The occurrence of soapstone on range V, lot 5, of Thetford township, Megantic county, near Robertson, has been mentioned in the Mines Branch Summary Report for 1919, p. 20. This locality was again visited in 1921.

The band of soapstone on the above lot is exposed in some small pits opened in the side of the ridge forming the north side of the valley, through which runs the Quebec Central railway. These pits have disclosed a band of friable, greenish schistose talc or soapstone, having a width of 12 feet. The material is very soft, and breaks up into thin layers.

Material similar to the above has been discovered at other points along this ridge, and at about the same general level. These outcrops indicate the existence of a persistent band of talc, extending for several miles. The most westerly outcrop seen was at a point about 100 feet north of the main pit of the Federal Asbestos Company, at Robertson, and the most westerly that on the lot mentioned above. The distance between these two points is about 2 miles.

About midway between the above outcrops there are some small workings, from which about 100 tons of material are reported to have been taken in 1920, for shipment to the United States.

The talc body at all these points measures from 12 to 15 feet in width, and dips 45° to 60° with the slope of the hill. The enclosing rock is variously slate, quartzite, or serpentine. The material is very uniform in character, and contains no gritty impurities. The surface material is too soft and friable to cut into blocks or slabs, but this characteristic may change with depth.

Diamond drilling of a talc deposit on range IV, lots 17 and 18, and range V, lot 18, township of Potton, Brome county, near Knowlton, was conducted in 1920 by the Talc Development Company of Canada, and the results are stated to have proven a promising body of talc. No further operations have been conducted to date.

CALCITE

Ontario

A very interesting and unusual occurrence of calcite occurs on concession VII, lot 4 W. ½ of the township of Palmerston,Frontenac county. The deposit lies 2 miles distant by winter road from Robertsville station on the Kingston and Pembroke branch of the Canadian Pacific railway. It is controlled by T. B. Caldwell, Perth, Ont., who conducted development work during 1920 and 1921, and has quarried several hundred tons of calcite ready for shipment.

The calcite is coarsely crystalline, and of a white to cream shade. It is very uniform in grain. The deposit is in the form of a vertical vein or band, measuring 90 feet maximum width between walls, at the southeast end. The length of the deposit is not known accurately, but the band has been traced northwest for a distance of 850 feet from the point of attack at the southeast outcrop. At the west end the width between walls is 65 feet. The enclosing rock is a reddish grey, fine-grained granite gneiss.

While at first sight the deposit appears to be of vein form, closer examination indicates that it may constitute a down-faulted block of limestone, which has suffered re-crystallization. This view is based on the following grounds: the material does not possess the highly spathic character and perfect cleavage usually exhibited by vein calcite; the deposit is remarkably uniform in character and composition over its entire width; there are no inclusions of country rock within the calcite body; there has been no alteration of the country rock adjacent to the calcite, the contact being very sharp.

Work done on the deposit consists of stripping at the southeast outcrop and quarrying back the face of the low ridge into which the calcite body runs.

An analysis of the calcite shows it to be very pure. The following analysis was furnished by the owner:—

Lime*	54·20
Silica	1·02
Iron oxide	0·24
Alumina	0·86
Magnesia	trace
Loss on ignition	43·50
Moisture	0·16
	99·98

*Equivalent to 96·75 per cent calcium carbonate.

It is hoped that the calcite may prove suitable for use in paints and as a filler or loader in rubber goods.

FELDSPAR

Ontario

A number of feldspar deposits were opened up during 1919-21, in the vicinity of Hybla, on the Trenton-Maynooth branch of the Canadian National railway. Most of the deposits proved to be small and were soon abandoned. The most important workings are those of the Verona Mining Company, on lots 18 and 19, concession VII, of the township of Monteagle. The quarry lies 2 miles from Hybla station, to which point the feldspar is hauled by wagon.

There is no machinery or plant on the property, drilling being done by hand. Operations to date (November, 1921) consist of considerable stripping on the crest and flank of the hill on which the deposit is situated, and a number of shallow pits have been opened. The chief opening is a pit 350 feet long by 30 feet wide, and averaging 15 feet deep. This pit exposes a rather irregular body of pink feldspar and quartz. The feldspar occurs as large crystals, or aggregates of crystals, separated by masses of quartz. In portions of the dike the two minerals occur predominantly in graphic granite intergrowth.

The nature of the ore entails close cobbing, and there is a large proportion of waste, but the quality of the clear feldspar is high. The company ships to its own mill at Rochester, N.Y.

Quebec

A deposit of feldspar was worked during 1921 on lot 47, range VIII, in the township of Aylwin, Hull county, the operator being W. G. Masson, of Ottawa. The workings lie one-fourth of a mile east of the Gatineau Valley branch of the Canadian Pacific railway, midway between Aylwin and Marks stations.

The deposit consists of a dike of grey to white soda feldspar (albite) mixed with a large proportion of quartz. The dike can be traced by isolated outcrops for a considerable distance, but the workings are all situated at

the most westerly exposure. This lies about 450 yards west of the railway, with which the quarry is connected by a tram line. Work was commenced early in 1921, and a considerable tonnage of feldspar was shipped. The whole of the output went to the Orford Soap Company, Manchester, Conn., for use in cleanser preparations.

An analysis of the feldspar of this deposit made in the Mines Branch laboratory showed it to contain 1·62 per cent of potash and 7·14 per cent of soda.

During 1921 considerable interest was evinced in the feldspar deposits of the Buckingham district, in Labelle county. The development, during the year, of a large body of very high grade feldspar, by M. J. O'Brien, Ltd., on lot 8, range I of the township of Derry, has led to active prospecting in the adjacent territory, and a number of deposits have been discovered. Many of these occurrences will doubtless undergo development during 1922, and shipments have already been made from one or two deposits.

The property now being operated by M. J. O'Brien, Ltd., lies nine miles north of the town of Buckingham, Que., and two miles from the Lievre river. In summer, the feldspar is hauled by wagon to the river, and loaded into scows, which are towed down to the railway at Buckingham. In winter transportation is by sleigh on the river ice to the same point.

A large, well-equipped camp has been established on the O'Brien property. A small portable boiler supplies steam for two steam drills.

Feldspar is won from a single large opening. This is an open cut carried 130 feet into the upper portion of the ridge crossed by the deposit, and having a width of 35 feet, and about the same depth. The actual feldspar dike is somewhat wider than 35 feet, and averages about 50 feet. In mining, however, a band along the walls, consisting of a somewhat lower grade of spar, is left standing, to be taken down separately. It is intended to work this deposit—which has been proved for a length of 300 feet—by carrying in a series of benches at successive levels, and working out the feldspar the length of the deposit at each level. In this way, no hoisting is necessary, the feldspar and waste being run out of the cut by tram.

This deposit is remarkable for the unusually clean feldspar carried. An unusually small proportion of quartz is present in the dike, and this quartz occurs as segregations between large feldspar crystals, and can be readily removed separately. The accessory minerals so often present in feldspar dikes, such as tourmaline, hornblende, mica, garnet, etc., are virtually absent. The feldspar is predominantly of a light cream colour, and much of it is remarkable for its vitreous appearance and partial translucency. These are characteristics of the feldspar of many of the deposits recently uncovered in the same district, and in some cases, the colour is almost a pure white, the feldspar then having a remarkable milky appearance.

Analyses[1] of two samples of the shipping grade of feldspar from the O'Brien property yielded:—

Silica	65·09	65·80
Alumina	18·85	19·74
Ferric oxide	0·03	0·03
Lime	0·21	0·11
Soda	2·11	1·74
Potash	13·42	12·32
	99·71	99·74

It seems probable, in view of the encouraging results obtained by M. J. O'Brien, Ltd., on the above property, and the number of deposits carrying a similar grade of spar that have been located in the same district, that the Buckingham region will shortly become an important feldspar producing centre.

V

ALKALI DEPOSITS OF WESTERN CANADA

L. Heber Cole

F. M. MacNiven, Assistant

Introductory

The search for potash in western Canada during the years of the great war led to the staking of claims on many of the alkali lakes and sloughs which occur in numerous localities in the morainic areas of the prairies, as well as in British Columbia, in the hope that potash salts would be present in commercial quantities. While the search for potash up to the present time, has been disappointing, the prospecting of the alkali deposits has demonstrated that western Canada possesses large reserves of sodium and magnesium compounds: such as sodium sulphate, sodium chloride, sodium carbonate, and magnesium sulphate.

Occurrences

Natural occurrences of soluble mineral salts are known in the provinces of Manitoba, Saskatchewan, Alberta, and British Columbia, either in the form of bedded deposits, or as brines. Some are of considerable extent, and are probably of sufficient size to warrant commercial exploitation, provided economical methods of recovery can be developed, and sufficient markets established.

The occurrences of these salts may be broadly classed under two types:—

(1) Solid salts and brines in undrained or partially drained basins.
(2) Brines of flowing streams or springs.

[1] Furnished by the operator.

Type (1).—Those of the first class are very numerous in the prairie provinces, and are, generally, similar in character, although the percentage of the different salts varies in different localities. In many cases the name alkali lake has been appropriately applied to deposits of this nature, since in the early spring, and often into late summer, the deposits are covered with water. The water accumulating from the melting snow and rain is often a foot or two in depth, and carries a considerable quantity of the alkali salts in solution, due to the dissolving of the top layers of the crystallized salts. In the late summer, especially when the season is a dry one, these so-called lakes become deposits of white alkali, which when seen from a distance resemble snow covered basins. A day's rain, however, or a rise in temperature, which increases the solubility of the salts, will quickly cover the surface of the crystals with a saturated brine. This brine will, sometimes, entirely disappear during one night, if there is a fall in temperature.

The deposits vary greatly in size, some being many acres in extent, and ranging up to 15 feet or more in thickness. The salts are generally found interbedded or mixed with calcareous mud and peaty material, and in only a few instances are the deposits in a pure enough form to be commercially marketable in their crude state. In many cases the salt crystals contain included mud stringers, so that the salts would require to be redissolved, and the impurities allowed to settle out of the solution before a pure product could be obtained. The mud beds also contain numerous crystals of the alkali salts.

Type (2).—Brine streams or springs occur in many places, and may carry sufficient salts in solution to warrant their commercial exploitation for medicinal and other purposes. In some of the occurrences of this nature the principal salt present is sodium chloride, the other salts being present only in small quantities. The brine springs of northern Manitoba are good examples of this class of deposit.

Composition

The composition of the salts occurring in these basins consists chiefly of mixtures of sodium and magnesium sulphates, in varying proportions, with, generally, small quantities of sodium chloride, sodium carbonate, calcium carbonate, and calcium sulphate, etc.

SODIUM SULPHATE

Sodium sulphate in the hydrous form (*known as Mirabilite or Glauber's Salt*) has the following composition:—

$$Na_2SO_4 + 10H_2O = \begin{cases} Na_2O & 19{\cdot}3 \text{ per cent (Soda)} \\ SO_3 & 24{\cdot}8 \text{ per cent (Sulphur trioxide)} \\ H_2O & 55{\cdot}9 \text{ per cent (Water)} \end{cases}$$

In its pure state it is white, transparent to opaque; and has a hardness of 1·5 to 2, with specific gravity 1·48. It is readily soluble in water, and at first is cool to the taste, and afterwards saline and bitter.

Sodium sulphate in the anhydrous form (*known as Thenardite*) has the following composition:—

$$Na_2SO_4 = \begin{cases} Na_2O & 56{\cdot}3 \text{ per cent (Soda)} \\ SO_3 & 43{\cdot}7 \text{ per cent (Sulphur trioxide)} \end{cases}$$

Its colour, when pure, is white, translucent to transparent, and the mineral has a hardness of 2 to 3, with specific gravity 2·68.

MAGNESIUM SULPHATE

Hydrous magnesium sulphate (*known as Epsomite or Epsom Salts*) has the following composition:—

$$MgSO_4+7H_2O = \begin{cases} MgO & 16{\cdot}3 \text{ per cent (Magnesia)} \\ SO_3 & 32{\cdot}5 \text{ per cent (Sulphur trioxide)} \\ H_2O & 51{\cdot}2 \text{ per cent (Water)} \end{cases}$$

This is a soft, white or colourless mineral, readily soluble in water and with a bitter saline taste. Its hardness is from 2 to 2·5 and specific gravity 1·75.

SODIUM CHLORIDE

Sodium chloride (known as *Halite or Common Salt*) has the following composition:—

$$NaCl = \begin{cases} Na & 60{\cdot}6 \text{ per cent (Sodium)} \\ Cl & 39{\cdot}4 \text{ per cent (Chlorine)} \end{cases}$$

The natural salt is nearly always impure. It has a hardness of 2·5, and specific gravity 2·1 to 2·6. It is colourless or white when pure, but often yellowish, or red, or purplish, due to the presence of metallic oxides, or organic matter. It is readily souble in water, and has a saline taste.

SODIUM CARBONATE

Hydrous sodium carbonate (known as *Natron*) has the following composition:—

$$Na_2CO_3+10H_2O = \begin{cases} Na_2O & 21{\cdot}7 \text{ per cent (Soda)} \\ CO_2 & 15{\cdot}4 \text{ per cent (Carbon dioxide)} \\ H_2O & 62{\cdot}9 \text{ per cent (Water)} \end{cases}$$

This is very soluble in water, from which it crystallizes as such only below 20° C. It has specific gravity 1·4 to 1·7, and hardness 1 to 1·5.

With these salts may be associated other soluble salts, and in small quantities, salts of the potassium and alum groups.

Uses

Sodium sulphate in the anhydrous form is more commonly known by its trade name Salt Cake. As salt cake, it finds its chief use in the manufacture of sulphate pulp; in metallurgical work in the refining of nickel; in the manufacture of window, plate, and bottle glass; and in making water glass. In the hydrous form, it is marketed as Glauber's Salt, and as such, is used in tanning; in the textile industry as a mordant; and in medicine.

Hydrous magnesium sulphate or Epsom Salts is largely used in the cotton trade for warp-sizing; it is also employed for medicinal and agricultural purposes, and in dyeing with aniline colours, since goods thus dyed are found to withstand better the action of soap.

Sodium chloride is the ordinary Common Salt of commerce, and as such, has many uses.[1]

Sodium carbonate in the anhydrous form, known under the trade name of Soda Ash, is one of the principal forms in which sodium is used in the alkali industry, since it frequently forms the base from which other sodium compounds are made. It is used extensively in the manufacture of glass, soap, and dyes, as well as in cleansing preparations, and tanning. In the hydrous form it is marketed under the trade names of Sal Soda, Washing Soda, or Crystal Carbonate, and is used in softening water and to replace soda ash when purity is essential. It is also used in cleansing compounds, or alone as washing soda. Sodium bicarbonate, or acid sodium carbonate, commonly known as Baking Soda, is generally marketed in a very pure form, and finds its principal use in baking.

Methods of Examination

The alkali deposits of western Canada are very numerous, and are scattered over a wide territory. On account of the great similarity of many of the deposits, it was thought best to confine detail work to several of the more typical deposits as representative of a district, and to obtain general data on the others.

Field Work

The field work on the deposits where detailed work was done consisted of:—

(1) Transit survey of the deposit and surrounding area, with contours at 10-foot intervals, sufficient to show the presence or absence of drainage to or from the deposit; also the location of springs and wells which might have any bearing on the source of the deposit.

(2) Detailed surface examination of the area included in the survey to determine the nature of soil and rocks of the surrounding country.

(3) Core drilling at regular intervals over the whole deposit, to determine its extent, depth, and composition.

(4) Sampling of all springs and wells in the area, and representative samples from all the cores obtained in the drilling operations.

(5) Collection of all data having any bearing on alkali deposits.

(6) Chemical analyses in the field, to quickly determine the quality of material in any deposit.

The field work on other deposits not studied in detail, consisted in visiting such deposits and obtaining representative samples from test pits, as well as brine samples from any springs feeding such deposits. In each case, general estimates of the size of such deposits were made from the township plans, and all available data collected.

[1] For details relative to the uses of Common Salt the reader is referred to "The Salt Deposits of Canada and the Salt Industry." No. 325, Mines Branch, Dept. of Mines. L. Heber Cole, 1915.

Apparatus and Methods

In surveying the lakes, a small light transit was employed, and measurements made with stadia. The elevations were determined by means of hand level.

The drilling was done with a power drill, built especially for this investigation, by the E. J. Longyear Company of Minneapolis, Minn., U.S.A. It consists of a type BD diamond drill, together with a hoist and a 2-inch by 3-inch Gould triplex pump, all operated by an 8 horse-power Cushman gasoline engine and mounted on a platform. The platform is bolted to a steel frame on wheels, so that the whole outfit is easily moved from place to place.

The entire weight is approximately 3,600 pounds.

In place of using diamonds for the bits, pieces of hardened steel with chisel edges were placed in the bit, and these were found to give good results. On account of the friable nature of the material being drilled, great care had to be taken in order to obtain satisfactory cores, since excessive vibration or too high a speed tended to break up the core into individual crystals. A saw tooth bit was tried, but it tore the crystals apart, instead of cutting them.

The drill is adapted to recover 1 inch and 2 inch cores.

A 200-foot steel cable attached to the hoist drum enabled the drill to be moved from place to place on the deposit, by its own power. When moving from one deposit to another the drill was hauled behind a Ford truck.

A complete chemical outfit for making preliminary analyses of samples in the field was included in the equipment. Aside from a standard Becker balance, Ralston still, and two coal oil burners, which were carried separately, the entire chemical apparatus was packed in two cases, with trays so arranged that any article could be taken out without having to remove all the apparatus. These boxes weighed, approximately, 300 pounds, and were shipped by freight to the west from Ottawa, and carted around from place to place without any breakage.

This outfit was sufficient for the determination of the principal salts in the lakes, and the composition of any deposit was quickly ascertained in the field. This knowledge was of great value in eliminating those lakes whose analyses showed them to be of no immediate commercial importance.

Description of Deposits

The following brief descriptions of deposits are of a preliminary nature, and only a few are given as typical of many examined.

Some of those visited, such as the large lakes south of Ceylon, Sask., and the one at Horizon, Sask., were covered with a considerable depth of water at the time, consequently little information was available.

Frederick Lake, Expanse, Sask.

This lake, situated in parts of sections 20, 21, 28, 29 and 33, township 12, range 28, W. 2nd meridian, was studied in detail, in order to obtain information relative to the nature of such deposits as would be applicable to all deposits of this character.

Frederick lake occupies a depression between a number of morainic ridges of the Missouri couteau, and covers an area of, approximately, 825 acres. It has a solid crystal bed composed principally of Glauber's Salt, with a small percentage of Epsom Salts.[1] It is separated from Johnson lake, on the north, by a narrow ridge composed of sand and gravel, which rises in places 30 to 40 feet. The level of Frederick lake is, approximately, 3 feet lower than Johnson lake. There is no apparent outlet, and some of the adjacent morainic ridges rise to a height of over 100 feet above the level of the crystal bed. Three springs feed into the depression from the shores. Analyses of these springs are given elsewhere in this report. Springs Nos. 1 and 2 are very similar in composition. Spring No. 3 varies considerably from the other two, but is very similar in composition to the water in Johnson lake. Inasmuch as this spring is situated on the shore of the lake nearest Johnson lake, it is quite probable that it is the result of seepage from the larger lake.

It is probable that at one time this lake was a bay of Johnson lake, since there are strong indications of beaches on the morainic slopes surrounding the lakes, at an elevation of from 30 to 40 feet above the present level. A rise of 40 feet in the present level of the water of the lake would submerge the narrow neck of land separating the two lakes, under about 10 feet of water. Conditions would, therefore, be favourable in this particular instance for the deposition of salts in the present basin of Frederick lake as a partially entrapped arm of the larger body of water of Johnson lake.

A series of some 65 holes were drilled over this deposit, and it was found that the hard crystal bed averaged 4·2 feet in depth. Beneath the crystal bed was a varying thickness of silty mud with salt crystals interspersed through it, and resting on a compact stoneless and highly calcareous clay. One typical hole drilled gave the following results:—

0 — 5′ 6″ hard, compact crystals.
5′ 6″—10′ 0″ soft mud and crystals.
10′ 0″—15′ 0″ silty clay, calcareous (stoneless).
15′ 0″—20′ 0″ coarse sand, calcareous bond.
20′ 0″—28′ 0″ boulder clay.

The materials comprising the surrounding country are boulder clay, gravel, sand, and silty clay.

The Canadian Pacific railway branch line from Moose Jaw to Assiniboia, Sask., runs along the south shore of this lake. A siding 200 yards long has been built at the lake.

Corral Lake, Sask.

This lake, situated on sections 14 and 23, township 14, range 20, west 3rd meridian, is, approximately, 12 miles northwest of the town of Gull Lake, Sask.

The deposit lies in an undrained depression in the eastern edge of the great belt of sand hills extending along the north side of the main line of the Canadian Pacific railway, between Swift Current and Maple Creek.

[1] Analyses of material from this deposit are given elsewhere.

The surrounding hills at this deposit are composed entirely of sand, no boulder clay or gravel area of any extent being noticed.

The deposit covers an area of 60 acres, and 15 test holes proved an average crystal bed of 5 feet.

There is one spring feeding this lake at the south end.

Alkali Lake near Inglebright, Sask.

This deposit occupies parts of six sections and is situated in sections 13 and 14, 23 and 24, 25 and 26 of township 16, range 25, west of the 3rd meridian.

By road, it is 40 miles almost directly north of Maple Creek, Sask., the nearest point on the railway.

The lake bottom is covered with a thick bed of hard crystals.

There are several springs along the shore, feeding into the lake, and in the deposit itself there are numerous springs coming up through the crystal bed. Some of these springs are 6 feet and 8 feet in diameter. When the deposit was visited October 24, 1921, there was one foot of water over the surface of the whole lake. This surface water was a saturated solution of sodium and magnesium sulphates.

There are 17 claims staked on this lake, under the Quartz Mining Regulations.

The impurity in the salts consists of sand rather than silt, such as is found in other deposits examined. One can walk onto the lake at almost any point, for the crystal bed is firm right to the shore. The shore is very sandy, and the banks, in most cases, are steep, rising from 10 to 40 feet above the lake level at a distance of 50 feet back from the shore.

The most noticeable feature of this lake, apart from its size, and the large amount of crystal present, is the numerous springs in the crystal bed at the north end of the lake. Many of these springs are bubbling up, and constantly depositing crystals in cones. It is said that in dry weather these springs build up cones, in places, to a height of 15 feet. Even with a foot of water on the surface of the crystals, these cones were noticed in the process of formation, projecting a few inches out of the water. No data were obtained as to the depth and tonnage of salts in this deposit, but it is evident that it is quite large.

Fusilier Deposit

(On the Lacombe-Kerrobert branch of the C.P.R.)

This deposit, owned by the Soda Deposits Ltd., Calgary, Alta., is situated on section 17, township 34, range 26, W. 3rd meridian, on Plover lake. Two claims of 40 acres each are located on this lake, taking in the whole area of the lake. When visited on Oct. 25, 1921, there was about one foot of water over the whole surface of the lake.

There are a number of springs feeding this lake, and the depth of the clean crystals on the top will average about 3 feet. The salts are practically pure sodium sulphate.

This deposit is being operated in a small way, and several thousand tons of clean salts have been excavated. These salts have been piled on the shore, and are gradually being dehydrated in drying sheds which have a capacity of 5 tons every two weeks. A service truck is being used to haul the material 5½ miles to the Canadian Pacific Railway station at Fusilier, Sask., where a warehouse 24 feet by 40 feet has been erected.

Senlac, Sask.

An interesting lake was examined on sections 19 and 30, township 39, range 25, west 3rd meridian. This lake is 12 miles north of Denzil, Sask., and 8 miles east of Evesham, Sask. The area of the lake is 185 acres, and, unlike many of the other lakes examined, the salt present is sodium chloride. The lake is fed by numerous springs, and a company known as the Senlac Salt Co., Calgary, Alta., has done considerable development work. The springs, which are very numerous, occur over various parts of the lake, and the degree of saturation of the water in the lake varies at different times of the year.

The company operating this lake built wooden cribbing around a number of the springs, and pumped the brine to shallow mud vats built on the shore. The solar system of evaporation was employed, and when the evaporation was completed the salt was shovelled onto drying boards, and allowed to drain, after which it was bagged and sold locally. There is also a small plant with two open pans operated by direct heat supplied by coal fires.

When the property was visited on Oct. 26, 1921, the whole plant was idle.

Two samples of the salt, furnished by the company, were analysed:—

	No. 1 coarse	No. 2 fine
Na	38·13%	36·87%
K	none	none
Mg	0·23	0·45
Ca	0·05	0·18
SO_4	0·08	0·35
Cl	59·43	58·14
Insol. in water	0·34	0·09
Loss at 110° C	1·02	2·24
Loss above 110° C	1·18	2·00
	100·46	100·32

A. Sadler, Analyst, Mines Branch.

Whiteshore Lake

This lake is situated in the northern part of township 36, ranges 16 and 17, west 3rd meridian. It is close to the stations of Oban and Palo on the main line of the Canadian Government railway, and Naseby, on the Winnipeg-Edmonton line of the Canadian Pacific railway. Its area is, approximately, 9 square miles.

When this deposit was visited, October 26, 1921, there was a foot of saturated brine over the whole surface of the deposit.

The clear surface deposit of crystals varied from a thin film to over 8 inches, underneath which is a thin layer of mud. Beneath the mud there is another bed of hard crystals of varying thickness, mixed with considerable mud.

An average percentage composition of the deposit, as calculated from a number of analyses, is shown in Table No. I, column 2.

Muskiki Lake (Dana, Sask.)

Muskiki lake, situated in townships 38 and 39, ranges 26 and 27, west of the 2nd meridian, contains large quantities of sodium and magnesium sulphates, as well as small percentages of other salts. The area of this lake is, approximately, 4,600 acres.

The property is owned by Salts and Chemicals, Ltd., Kitchener, Ont., and this company has done a large amount of experimentation on the recovery of the salts in a commercial form, both at their plant at the lake and at their refining plant at Kitchener, Ont.

The country surrounding this lake is rolling prairie, and the lake is the lowest point for some miles around. There is no apparent outlet, and there are numerous springs feeding into the lake around the shores and in the lake itself. One group of springs was very noticeable, about 10 feet from the shore on the east side, bubbling up and keeping the surface water in constant agitation. The shore at this point is built up of calcareous sinter intimately mixed with an ochre. The gravelly material making up the beach shingle for quite a distance on the south side of this point is made up of nodules of tufa in varying sizes. These materials are probably deposition products from the springs.

The deposit of salts in the bed of this lake varies greatly in thickness from place to place. It is composed chiefly of Glauber's Salts, with an average of 10 per cent of Epsom Salts. The lake brine, on the other hand, runs high in magnesium sulphate and also carries a small percentage of potash salts.

The crystallized salts in the bed of the lake are harvested as such, when conditions are favourable, and stored in drainage sheds at the lake. When sufficiently drained they are shipped to Kitchener, where the refining of the salts for the market is accomplished. The brine is evaporated in the plant at the lake, from whence the crude salts are shipped to Kitchener, for the recovery of refined Epsom Salts, as well as magnesium carbonate and potash salts.

Chemical Analyses

A number of analyses given below show the wide variation in the composition of the lakes visited.

Frederick lake, about 3½ miles from Expanse, Sask., was examined first, and a number of samples from various parts were taken. The deposit of crystals varied in thickness from a few inches at the edge to about 5 feet in the middle. Core samples were taken, and an analysis was made of each foot, thus giving an idea of the uniformity of the bed. Some of the results are given below.

Hole No. 3—Frederick Lake, Expanse, Sask. Crystal bed 3 feet thick.

——	1st foot	2nd foot	3rd foot	[1]Composite
Insoluble	0·78	3·53	1·49	1·86
Iron oxide and alumina	0·06	0·20	0·09	0·20
Calcium carbonate	1·70	5·85	2·10	2·17
Magnesium carbonate	0·76	3·70	2·70	1·62
Magnesium sulphate	2·63	2·17	4·01	3·30
Sodium chloride	0·61	0·46	1·39	1·02
Sodium sulphates	92·80	83·65	88·09	89·28
Total	99·34	99·56	99·87	99·45

Hole No. 1—Crystal bed 4 feet thick

——	1st foot	2nd foot	3rd foot	4th foot	[1]Composite
Insoluble	2·54	3·30	3·46	3·55	3·49
Iron oxide and alumina	0·12	0·42	0·21	0·18	0·15
Calcium carbonate	2·53	2·04	4·08	3·13	2·88
Magnesium carbonate	1·65	1·15	3·67	2·32	1·11
Magnesium sulphate	5·25	5·32	4·20	5·25	6·03
Sodium chloride	0·90	0·98	1·02	1·00	0·87
Sodium sulphates	86·06	86·38	83·06	84·52	85·29
Total	100·05	99·59	99·70	99·95	99·82

[1]The composite was not made up from the samples for each foot but was an independent sample through the crystal bed.

The water from several springs flowing into this lake was analysed, and the results are given below in parts per 1,000,000.

——	No. 1 spring	No. 2 spring	No. 3 spring
Insoluble	30·2	30·0	20·0
Iron oxide and alumina	7·9	3·6	5·6
Calcium carbonate	310·4	304·2	.....
Magnesium carbonate	209·8	242·0	180·6
Magnesium sulphate	.....	.....	1,222·0
Sodium chloride	5·14	8·8	.....
Sodium sulphate	968·8	925·8	4,395·0

Corral lake situated about twelve miles north of Gull lake, Sask.

[1]*Hole No. 1—on the shore*

——	1st foot	2nd foot	3rd foot	4th foot	5th foot
Insoluble	70·09	67·80	59·13	67·87	70·52
Iron oxide and alumina	1·83	1·60	1·50	0·42	0·63
Calcium carbonate	0·84	0·35	0·68	0·71	1·49
Calcium sulphate	.....	.....	.....	.....	2·07
Magnesium carbonate	1·36	.....	1·47	.....	.....
Magnesium sulphate	.....	1·94	2·11	1·83	3·50
Sodium chloride	0·60	0·50	0·40	0·55	0·81
Sodium sulphate	25·03	27·55	34·64	28·10	20·69
Total	99·75	99·74	99·93	99·48	99·71

[1]This hole was sunk to ascertain whether the crystal bed extended farther than was indicated on the surface.

Hole No. 3—Typical sample of the deposit

——	1st foot	2nd foot	3rd foot	4th foot	5th foot
Insoluble	14·74	8·04	9·73	9·11	11·80
Iron oxide and alumina	0·10	0·004	0·005	0·008	0·008
Calcium carbonate	2·22	0·86	0·95	0·76	0·86
Magnesium carbonate	4·01	2·06	2·13	2·10	1·53
Magnesium sulphate	.....	.....	.....	.....	1·24
Sodium chloride	0·29	0·25	0·25	0·25	0·29
Sodium sulphate	78·32	87·89	86·56	87·49	83·93
Total	99·68	99·10	99·62	99·71	99·66

This lake is surrounded by sandy land, which may be either washed or blown on to the deposit, hence the high percentage of insoluble material.

The analyses of crystal beds in several lakes are given in the columns of Table I, and of brines in Table II.

TABLE I

Constituents	Lake near Gladmar, Sask.	Oban, Sask. Whiteshore lake	Lake Alta.
	%	%	%
Insoluble	trace	0·1	14·03
Iron oxide and alumina	absent	.....	.....
Calcium carbonate	absent	.....	0·59
Calcium sulphate	.....	0·5	.....
Magnesium carbonate	absent	.....	1·10
Magnesium sulphate	.....	5·2	.....
Sodium chloride	0·12	1·3	0·10
Sodium sulphate	99·49	92·0	84·18

Column 1—Only pure crystals were taken in the analysis—E. A. Thompson, analyst.
Column 2—Average analysis made by Dr. Thorwaldson, Saskatchewan University.
Column 3—Analysis made by H. A. Leverin, Mines Branch, Ottawa.

TABLE II

Constituents	1 Parts per million	2 Parts per million	3 Parts per million	4 Parts per million	5 Parts per million
Organic matter	68·4	6·7	8·3	.....	477
Suspended matter	123·5	17·8	208·6	.....	.....
Silica	1·4	13·5	69·1	0·22	.....
Iron oxide and alumina	27·3	0·9	25·0	0·085	nil
Calcium carbonate	trace	121·9	trace	2·055	nil
Calcium sulphate	195·3	322·7	895·1	17·84	nil
Magnesium carbonate	501·3	192·3	309·6	0·186	146
Magnesium sulphate	197·7	.....	1,232·0	56·24	.....
Sodium chloride	58·3	15·4	96·7	17·84	823
Sodium sulphate	441·4	133·9	1,685·6	68·02	358
Potassium chloride	.....	.....	.....	1·63	.....
Sodium carbonate	.....	.....	.....	2·25	3,410

1. York lake, Melville, Sask.
2. A lake near Duff, Sask.
3. Water, Talmage, Sask.
4. Big Quill lake, Wynyard, Sask.
5. Soda lake, Maidstone, Sask.

Preparation for the Market

In preparing this product for the market, the guiding principle, of course, will be the specifications laid down by the consumer. There are several markets for sodium sulphate. The chief ones, or those which would probably take the large tonnages, are the paper industry, in which the salt is used in the manufacture of sulphate pulp, or, more correctly, sulphide pulp, and in the manufacture of glass.

As to the glass industry, we have been unable to get a copy of the exact requirements, but the chief ones are (1) that the material be dry; and (2) that the iron content be very low, possibly in no case above ·02-·03%.

A small percentage of insoluble material, if siliceous in nature, would not be objectionable, and small amounts of magnesia and lime would probably be permissible. It is to be remembered, however, that the soda base is the essential constituent, and that the materials mentioned above are to be regarded as impurities which may lower the effective value of the product.

Relative to the requirements of the paper industry, we are indebted to Dr. McLean of J. T. Donald and Company, Ltd., Montreal, for the analysis of a shipment of salt cake, which was accepted and used in the manufacture of sulphate pulp. He said that the analysis mentioned could be taken as a standard.

The following is the analysis of a shipment of salt cake obtained as a by-product in the manufacture of hydrochloric acid:—

	per cent
Moisture	0·24
Silica	0·19
Iron oxide	0·45
Alumina	trace
Sodium chloride	2·15
Free sulphuric acid	1·96
Sodium sulphate (calc.)	95·01

Less than 1 per cent sodium chloride is advisable, and not more than ·5 per cent iron; free sulphuric acid appearing in the analysis would not be found in the product of the alkali lakes. No allowance is made for calcium and magnesium sulphates. They would probably not prove detrimental to the product. It is thought that the magnesium salt would make the paper a little more flexible, although, perhaps, a little heavier. Possibly 5 per cent of each of these salts would be allowed, but so large an amount of impurities would, very appreciably, lower the value of the salt cake.

Keeping in mind, therefore, the specifications required, only those deposits which run approximately 90 per cent or over, sodium sulphate, in the dry state, should be worked at the present time.

The proximity of the deposit to a railway is also of the utmost importance, because of the cost of haulage.

Sodium sulphate, as it occurs in the lakes, is in the form of crude Glauber's salt. If the insoluble material (sand, organic matter, etc.) is high, it will have to be removed first. This may be done by bringing the salt into solution in water and then allowing the insoluble to settle out in settling tanks. The clear solution may then be decanted into crystallizing pans or tanks, and the salt allowed to crystallize out.

DRYING PROCESSES

There are a number of methods suggested for the dehydration of Glauber's salt. Some of these are given below.

Air Drying.—As is well known, Glauber's salt when exposed to the air loses its water of crystallization, and forms a fine white powder of anhydrous sodium sulphate. In this process, the hydrated crystals are exposed to the air, on trays with cotton bottoms which allow the air to circulate freely through the material. About two weeks is required to lower the moisture content from 56 per cent, approximately, to 15 per cent. The material is removed to a building where it is placed on trays, and the last traces of water are easily removed in a day or two by the heat of a stove.

This method is in operation at an alkali lake 5½ miles from Fusilier, Sask. The company has erected a building or shed about 10 feet high by 12 feet wide by 100 feet long. In this shed, the trays are arranged in rows, one above another, about two inches apart, and are filled with hydrated crystals. The trays are pushed along one following another until the shed is filled. The sides of the shed are built in sections or doors on hinges, which open from the top, and lie back on the ground. This allows the air to circulate above and below the exposed crystals, and thus carry off the water. About five tons of the dried material is obtained in this way,

in two weeks. Two men can easily fill or empty the shed in one day. In order, then, to have an output of five tons per day, fourteen sheds of the size stated above would be required. This would necessitate quite an initial outlay, and the cost of maintenance would be considerable. However, this method is probably the most economical that has been devised so far.

Direct Evaporation of the water.—One method is, to heat the salt in large open iron pans, and to expel the water by direct evaporation. There are two main objections to this method: (1) that the dried salt cakes solidly in the pans, and would have to be loosened with picks, which would increase the cost; and (2) the cost of fuel in heating.

Rotating dryers have been suggested for drying purposes, but whether these would work on material which goes into solution in its own water of crystallization when the temperature is raised is doubtful. At about 33° C., approximately 30 per cent of the sodium sulphate in solution is precipitated, and forms a rather pasty mass. Special experiments will have to be conducted in order to ascertain whether the material would cake in the dryer, and thus clog it.

Cement kilns with certain modifications have been used in drying solutions of potassium salts. At the upper or feed end of the kiln, the inside was lined with 2 inch by 4 inch scantlings, placed lengthwise and on edge about 2 inches apart, blocks being inserted between to hold them rigidly in place. Then, at the end of the scantlings, the kiln was lined with 2 inch by 4 inch scantlings placed side by side for about 6 feet. The heat was applied at the lower end by either pulverized coal, oil, or gas.

The solution was fed in a constant stream and ran into the spaces or troughs between the scantlings. As the kiln revolved, the solution was carried up and then poured out through the hot gases, finally falling to the bottom only to be taken up again. The solution becoming more concentrated all the time, gradually worked forward, until finally all the water was expelled, and only the salts were left. These issue at the firing end as in cement making. The inside of the kiln at the feed end was arranged with alternate troughs and complete bands, but this work was only carried to a point where the lining was not affected by the heat. Looking into the kiln from the feed end, the appearance was strikingly similar to a shower of rain.

Unless this method could be used when feeding Glauber's salt crystals, it is evident that it would be less economical than evaporation in open pans. Then, too, power would be required to rotate the kiln.

On account of the salt coming in direct contact with the fuel, the dried material may be contaminated with coal dust, if that is used for fuel.

Production and Market Conditions

Sodium Sulphate.—The production of sodium sulphate from the natural deposits of western Canada is still in its infancy. There is a small but fairly steady market for this material in the anhydrous form in the pulp and paper industry, but the consuming centres are in eastern Canada, and the present high freight rates from the deposits to the markets are an important factor. Very little of the crude material, as such, can be marketed, and this necessitates the installation of refining plants.

The product has to compete against that produced as a by-product from acid manufacture, but there are hopes that simple refining methods and lower freight rates will eventually help this industry.

The Salts and Chemicals Ltd., of Kitchener, Ont., operating Muskiki lake, Sask. (townships 38 and 39 ranges 26 and 27, west 2nd meridian), as already stated, have erected refining plants at their lake, and also at Kitchener, Ont., in which they have conducted much experimental work, and hope shortly to be in a position to place refined products regularly on the market. The Soda Deposits, Ltd., operating Plover lake, Sask., (township 34, range 27, west 3rd meridian), have been producing small quantities of high grade anhydrous sodium sulphate.

Frederick lake, near Expanse, Sask. (township 12, range 28, west 2nd meridian), will likely be a producer shortly, as a plant is now being installed by the Bishopric and Lent Co., of Cincinnati, Ohio. There are many deposits which could produce sodium sulphate, but only those close to the railways can be considered at the present time as prospective producers.

Practically all the salt cake so far used in the country has been obtained as a by-product from the manufacture of hydrochloric acid. The amount produced by this process in future will necessarily be governed by the market for hydrochloric acid.

Salt cake is being produced in Canada by the following firms:—

Grasselli Chemical Co., Hamilton, Ont. (by-product).
Nichols Chemical Co., Montreal, Que.
Plants: Sulphide, Ont. (by-product).
Capelton, Que. (by-product).
Canadian Explosives, Ltd., James Island, B.C. (by-product).
Salts and Chemicals, Ltd., Kitchener, Ont.
Plants: Kitchener, Ont. (natural).
Dana, Sask. (natural).
Soda Deposits, Ltd., Calgary, Alta.
Plant: Fusilier, Sask. (natural).

The Canadian production of salt cake and Glauber's salt as furnished by the Dominion Bureau of Statistics for 1918-1921 was as follows:—

	1918		1919		1920		1921	
	Tons	Value	Tons	Value	Tons	Value	Tons	Value
		$		$		$		$
Salt cake	6,001	133,544	3,197	57,045	5,524	111,983	2,418	54,804
Glauber's salt	2,358	60,281	1,423	45,731	1,781	50,336	1,239	42,719

Canadian imports of salt cake and Glauber's salt are as follows:—

Calendar Year	Salt cake		Glauber's salt	
	Amount	Value	Amount	Value
	lbs.	$	lbs.	$
1910	17,728,543	95,054	1,080,309	5,217
1911	13,782,241	88,761	1,531,555	7,826
1912	19,243,823	97,768	1,951,619	9,129
1913	25,902,190	133,030	811,053	3,815
1914	38,175,604	170,333	810,062	3,407
1915	30,970,231	147,047	840,994	8,058
1916	42,194,077	178,370	522,703	8,133
1917	71,583,645	560,711	722,913	16,248
1918	68,773,441	676,571	686,712	9,748
1919	47,905,004	343,007	738,423	9,763
1920	85,948,000	958,628	565,746	8,364
1921	54,379,450	680,083	579,928	4,521

Magnesium Sulphate.—There is a small production of magnesium sulphate from the natural deposits of western Canada, the main producing districts being, five lakes near Basque, B.C.; a lake near Clinton, B.C.; and Muskiki lake at Dana, Sask. A big deposit is located near Kruger mountain, B.C., and it is possible that some of the alkali deposits of Saskatchewan and Alberta may, in time, become producers.

The market for this material remains quiet. The producing localities are unfavourably situated as regards the present market in eastern Canada, hence there is still a considerable import, mainly into the markets of the east. The Canadian production, however, seems to have caused a noticeable reduction in the import figures for 1921.

The Canadian producers of magnesium sulphate are:—

The Basque Chemical Production Co., Ltd., Vancouver, B.C.
Plant: Basque Ranch (near Ashcroft, B.C.).

Stewart-Calvert Co., Inc., Oroville, Washington, U.S.A.
Plant: Clinton, B.C.

Salts and Chemicals, Ltd., Kitchener, Ont.
Plants: Kitchener, Ont.
Dana, Sask. (Muskiki lake).

The production, imports and exports of magnesium sulphate are as follows:—

	1917		1918		1919		1920		1921	
	Tons	Value	Tons	Value	Tons	Value	Tons	Value	Tons	Value
		$		$		$		$		$
Quantity extracted..	2,600	.....	4,500	.....	.....	.....	2,056	.....	.....	.....
[1]Quantity shipped...	929	4,645	1,949	14,565	738	9,115	1,125	28,137	614	20,961
Exports.....	.....	.....	.....	.....	.....	15	743	3,737	119	4,562
Imports.....	.....	.....	.....	.....		(a)54,779	.....	72,709	.....	29,987

(a) Not separately classified prior to April, 1919.
[1]Figures furnished by Dominion Bureau of Statistics.

Market Prices.—The market prices for alkalis are constantly varying. The following figures, as reported in the Oil, Paint, and Drug Reporter, New York, give the New York market prices for the years 1914 to date:—

—	Aug. 14, 1914	Jan. 1, 1915	Jan. 1, 1917	Jan. 1, 1918	* 1919	* 1920	* 1921
	$	$	$	$	$	$	$
Salt cake, ground—bbls. per ton	11·00 to 13·00	11·00 13·00	....	30·00 to 35·00	12·00 to 30·00	17·60 to 80·00	17·00 to 28·00
Glauber's salt—cwt.	0·65 to 0·75	0·60 to 0·75	0·60 to 0·65	0·90 to 1·00	1·00 to 2·25	1·15 to 2·65	1·00 to 1·75
Epsom salt, U.S.P—cwt.	Not quoted prior to 1918			3·62½ to 3·90	2·75 to 3·62½	2·50 to 5·50	2·20 to 2·75
Epsom salt, tech.—cwt.	1·00 to 1·10	1·75 to 2·00	1·75 to 1·85	3·37½ to 3·50	1·80 to 3·37½	1·75 to 3·50	1·10 to 1·75

*High and low figures for year.

VI

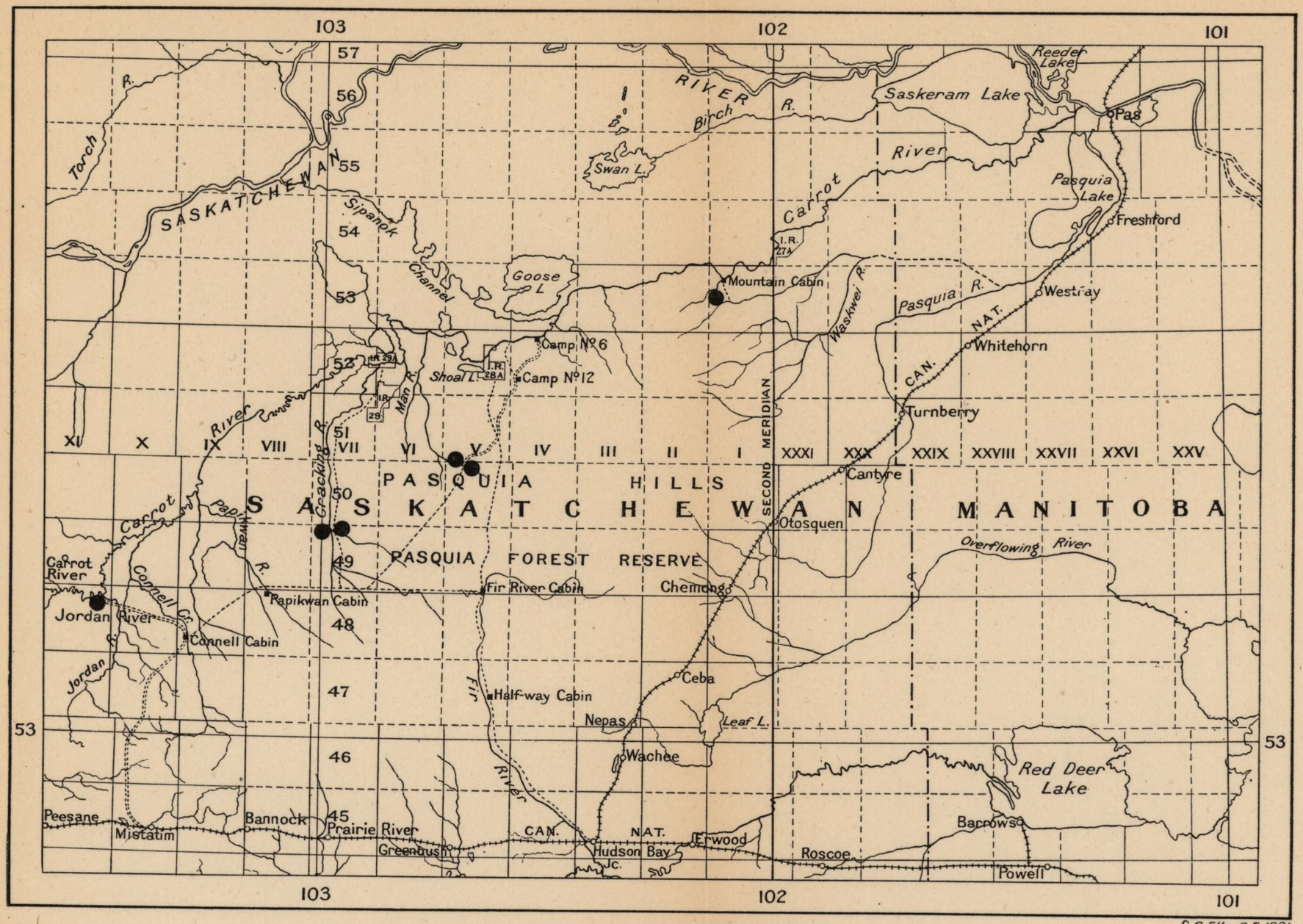
CRETACEOUS SHALES OF MANITOBA AND SASKATCHEWAN, AS A POSSIBLE SOURCE OF CRUDE PETROLEUM

S. C. Ells

INTRODUCTORY

During recent years attention has been directed to reported occurrences of oil shale in the provinces of Manitoba and Saskatchewan. The shales are exposed along the escarpment of the Pembina, Riding, Duck, and Porcupine mountains, which border the lake plain and Red River valley of Manitoba. They are also found in the escarpment of the Pasquia hills, a northern extension of the same series. These hills, dignified by the name of mountains, constitute the erosion escarpment of the Cretaceous beds which form the first prairie step. The eastern edge is indented by drainage valleys of varying importance, which separate the hill features into groups. The Pasquia hills, and Porcupine, Duck, and Riding nountains, occupy an area which is bounded toward the north and northwest by the Carrot river, and toward the east by waterways, which include Moose lake, Cedar lake, and lake Winnipegosis. Broad, low-lying, slightly undulating, lacustral plains, which formed the bed of glacial Lake Agassiz, stretch away from the various waterways of the lower slopes of the hills. These lower slopes are marked by a series of old lake beaches, and rise by easy gradients, through some five or six hundred feet, to the more abrupt escarpment of the main ridge.

The whole area is well watered by numerous small streams few of which have a width greater than 60 feet. In descending from the table lands, these streams, for the most part, flow with rapid current along boulder-strewn channels, deeply entrenched in precipitous valleys and ravines, where active erosion and landslides are much in evidence. On reaching the lower slopes of the hills, the current slackens, and many excellent geological sections are exposed in cut banks at concave bends.



S.C.Ells, C.E. 1921.

Map showing position of Outcrops of Cretaceous Shales sampled during year 1921, in Pasquia Forest Reserve, Saskatchewan.

Scale of Miles

10 5 0 10 20 30

FIG. 1.

SASKATCHEWAN MANITOBA
PASQUIA HILLS
PASQUIA FOREST RESERVE
Red Deer Lake
Pasquia River
Carrot River
Jordan River
Saskatchewan River
Hudson Bay
Prince Albert
Fort Pelly
Fort Qu'Appelle
Regina
Moose Jaw
Swift Current
Medicine Hat
Calgary
Edmonton
Saskatoon
Prince Albert
Winnipeg
Brandon
Portage la Prairie
Dauphin
Flin Flon
The Pas
Thompson
Churchill
Hudson Bay
Port Nelson
Fort Churchill
Fort Severn
Fort Albany
Moose Factory
Cochrane
Timmins
Sudbury
Kenora
Fort Frances
Port Arthur
Fort William
Sault Ste. Marie
Owen Sound
Toronto
Hamilton
Kingston
Ottawa
Montreal
Quebec
Sherbrooke
Moncton
Saint John
Fredericton
Halifax
Sydney
Charlottetown
Prince Edward Island
Newfoundland
St. John's
Labrador
Ungava Bay
Hudson Strait
Baffin Island
Frobisher Bay
Cape Dorset
Lake Harbour
Pangnirtung
Broughton Island
Clyde River
Pond Inlet
Arctic Bay
Resolute
Grise Fiord
Eureka
Alert
Ellesmere Island
Axel Heiberg Island
Ellef Ringnes Island
Amund Ringnes Island
Meighen Island
Cornwallis Island
Devon Island
Somerset Island
Prince of Wales Island
Victoria Island
Banks Island
Melville Island
Bathurst Island
Byam Martin Island
Prince Patrick Island
Mackenzie River
Great Slave Lake
Great Bear Lake
Yellowknife
Fort Simpson
Fort Providence
Fort Smith
Hay River
Fort Resolution
Fort Liard
Fort Nelson
Fort St. John
Dawson Creek
Peace River
Grande Prairie
Whitehorse
Dawson
Mayo
Keno
Stewart
Watson Lake
Teslin
Carcross
Atlin
Juneau
Ketchikan
Prince Rupert
Vancouver
Victoria
Nanaimo
Kamloops
Penticton
Kelowna
Revelstoke
Cranbrook
Fernie
Crowsnest Pass
Lethbridge
Medicine Hat
Brooks
Drumheller
Red Deer
Camrose
Wetaskiwin
Leduc
Stony Plain
Jasper
Banff
Lake Louise
Field
Golden
Salmon Arm
Vernon
Merritt
Hope
Chilliwack
New Westminster
Nanaimo
Courtenay
Campbell River
Port Hardy
Queen Charlotte Islands
Haida Gwaii
Skeena River
Nass River
Stikine River
Liard River
Mackenzie Mountains
Rocky Mountains
Cassiar Mountains
Coast Mountains
Selkirk Mountains
Purcell Mountains
Columbia Mountains
Cariboo Mountains
Omineca Mountains
Peace River
Athabasca River
North Saskatchewan River
South Saskatchewan River
Assiniboine River
Red River
Saskatchewan River
Churchill River
Nelson River
Winnipeg River
Lake Winnipeg
Lake Manitoba
Lake Winnipegosis
Lake Athabasca
Great Slave Lake
Great Bear Lake
Lake Athabasca
Reindeer Lake
Cree Lake
Wollaston Lake
Lake Athabasca
Lake Athabasca
Lake Athabasca

Through the low-lying alluvial lands, the banks, as a rule, are low, and the channels of the meandering type.

Outcrops of shales examined on Birch river, Favel river, Sclater river, Pine river, Vermilion river, and Ochre river, are all near the Winnipeg-Prince Albert branch of the Canadian National Railways, and may be easily reached by highway roads. Outcrops examined on the Tee river—a branch of Pasquia river, Man river, Cracking river, and Papikwan river in the Pasquia hills, are somewhat remote from rail transportation.

Toward the east and northeast, the slopes of the Pasquia hills are marked by somewhat sharp gradients. In passing westward, however, the slopes become more gentle, and rock exposures along the various streams less frequent. The elevation of the summit has not been accurately determined, but apparently ranges from 2,000 to 2,300 feet above sea level datum. The area throughout is densely wooded. Apart from a number of comparatively limited areas reserved as timber berths, the forest growth consists, principally, of large poplar and birch. Much of the land is of excellent quality, and, when cleared, should prove well adapted to agricultural development.

Outcrops of shale along streams which drain the northeastern slopes of the hills (as on Tee river and Man river) can best be reached from the Pas. From Mountain cabin on Carrot river, some 60 miles W.S.W. from the Pas, a good summer trail, approximately 2½ miles in length, leads south to Tee river. From Camp No. 6 on Carrot river, some 90 miles from the Pas, a fair bush road, approximately 17 miles in length, leads to Man river.

Papikwan and Cracking rivers can best be reached from McDonald's siding, 2½ miles west of Mistatim station, on the Canadian National Railways. From McDonald's siding, a fair wagon road, some 24 miles in length, leads to Connell cabin. From Connell cabin to Papikwan cabin, a distance of approximately 12 miles, a fair pack trail is available; but between Papikwan cabin and the shale outcrops at the forks of Cracking river, the trail is wet and difficult. Other trails indicated on the accompanying map are, for the most part, poor, and in many instances, difficult to follow.

GEOLOGY

The general geology of the area was worked out many years ago by Tyrrell, Dowling, and McInnes, and the reports then issued[1] remain authoritative. The general geological features are simple, and may be very briefly summarized as follows:—

The Pasquia hills, and Porcupine, Duck, and Riding mountains, are built up of Cretaceous sediments, resting unconformably on limestones of Palæozoic age. Apart from type fossils, which are found at many localities, the principal subdivisions of the Cretaceous, within the area under consideration, may frequently be recognized by lithological characteristics.

[1] Report on Northwestern Manitoba with portions of the adjacent districts of Assiniboia and Saskatchewan, J. B. Tyrrell and D. B. Dowling, Geol. Survey of Canada, 1892.
The Basins of Nelson and Churchill Rivers, Wm. McInnes, Geol. Surv. of Canada, Mem. 30, 1913.

A marker band of impure limestone, averaging some four feet thick, and containing typical fossils, occurs near the top of the Niobrara, and apparently extends over a wide area. Other marker bands are of local significance only. In the following general section, estimated thicknesses are given, but, pending the collection of further data, these must be considered as approximations.

Cretaceous	Montana	Pierre (upper) Odanah shales	400 feet
		" (lower) Millwood shales	650 "
	Colorado	(Boyne shale) Niobrara (Morden shale) (Assiniboia shale)	130–240 "
		Benton shale	180 "
		Dakota sandstone.	
Devonian		Unconformity.	

The detailed geological structure has not been accurately worked out, although numerous exposed sections, together with correlated elevations, indicate that the general dip of strata of Cretaceous age is almost horizontal. Between Winnipegosis and the well drilled by the Manitoba Oil Company, 11 miles southwest of Dauphin, the dip of Upper Palæozoic limestones is apparently about four feet to the mile, in a southwesterly direction. South of Swan River valley, Cretaceous sediments also dip at a very low angle toward the southwest. North of this point, the strata flatten out, and finally dip toward the north. The same general horizontal structure apparently typifies strata underlying the Pasquia hills. Minor local folding was observed, but this in itself can scarcely be considered as of economic importance. The general physical characteristics of members of the Cretaceous system may be briefly summarized as follows:—

Pierre.—Odanah series. Dark fissile shales, usually poor in fossils, and weathering to light grey fissile flakes. Often rusty along joint planes.

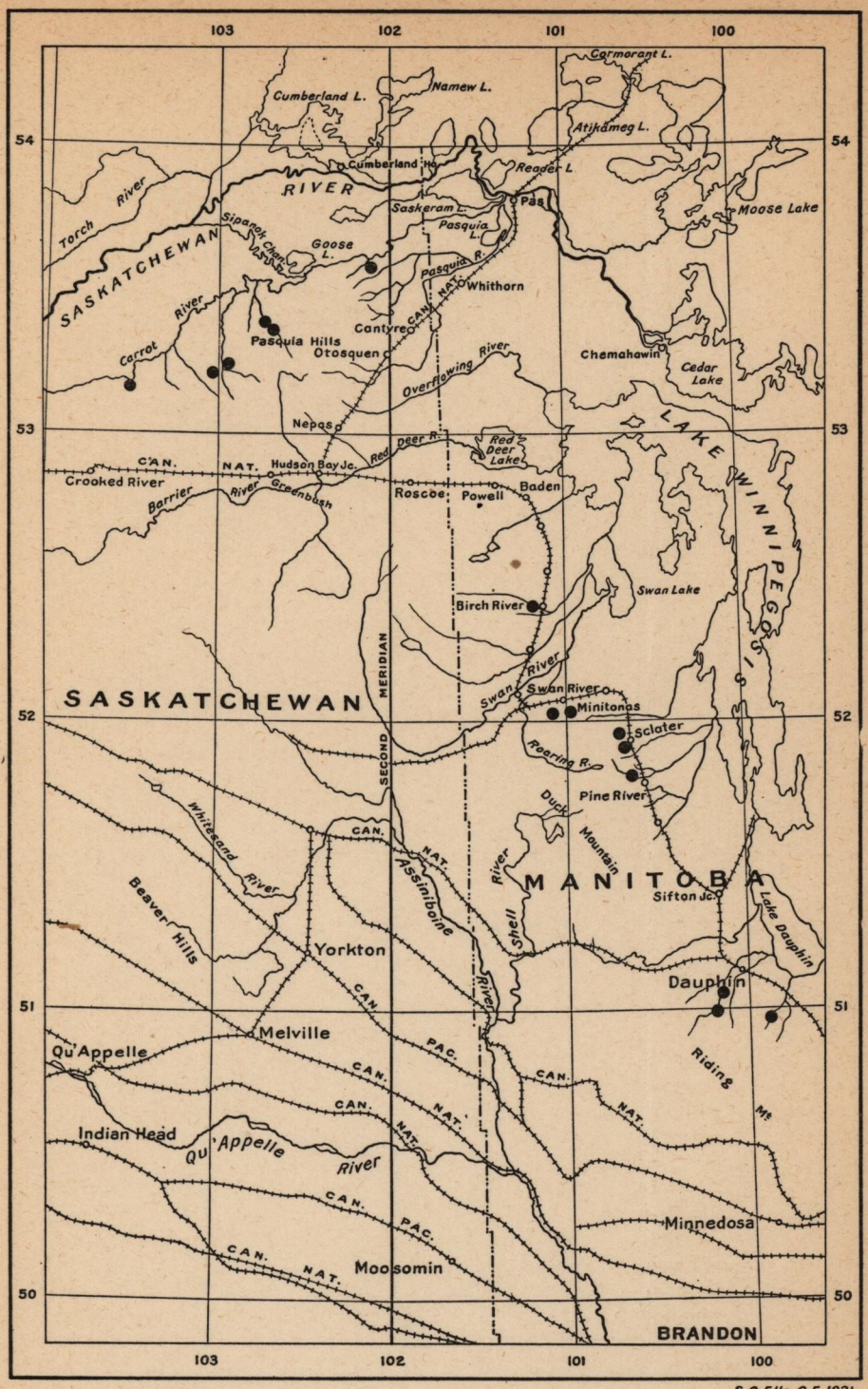
Millwood series. Soft grey shales, sometimes almost black in colour. Weathers to light grey flaky particles, which eventually disintegrate to dark plastic clay. Slopes adjacent to cut banks exhibit marked evidence of instability. Ironstone nodules are common, particularly near contact with Niobrara.

Niobrara.—Dark grey clay shales, frequently mottled, and interbedded with heavy bands of hard calcareous shale, usually highly fossiliferous, and almost black in colour. Shale weathers to light grey colour. Comparatively weather resistant.

Benton.—Very dark, soft, non-calcareous, somewhat carbonaceous, evenly bedded shale, poor in fossils. Weathers to small thin flakes, which rapidly disintegrate into dark plastic clay.

Dakota.—White, light grey or greenish sandstone. Usually somewhat soft, but in places having the hardness of quartzite.

With the exception of the upper and the lower members of the Cretaceous system, the sediments exposed within the area under consideration thus consist chiefly of rather soft, greenish grey, clay shales, more or less darkened owing to the presence of hydrocarbons and iron sulphides. Moreover, many of these shales, when freshly broken, emit a marked odour of petroleum, which, on exposure, rapidly passes off. When disintegrated by



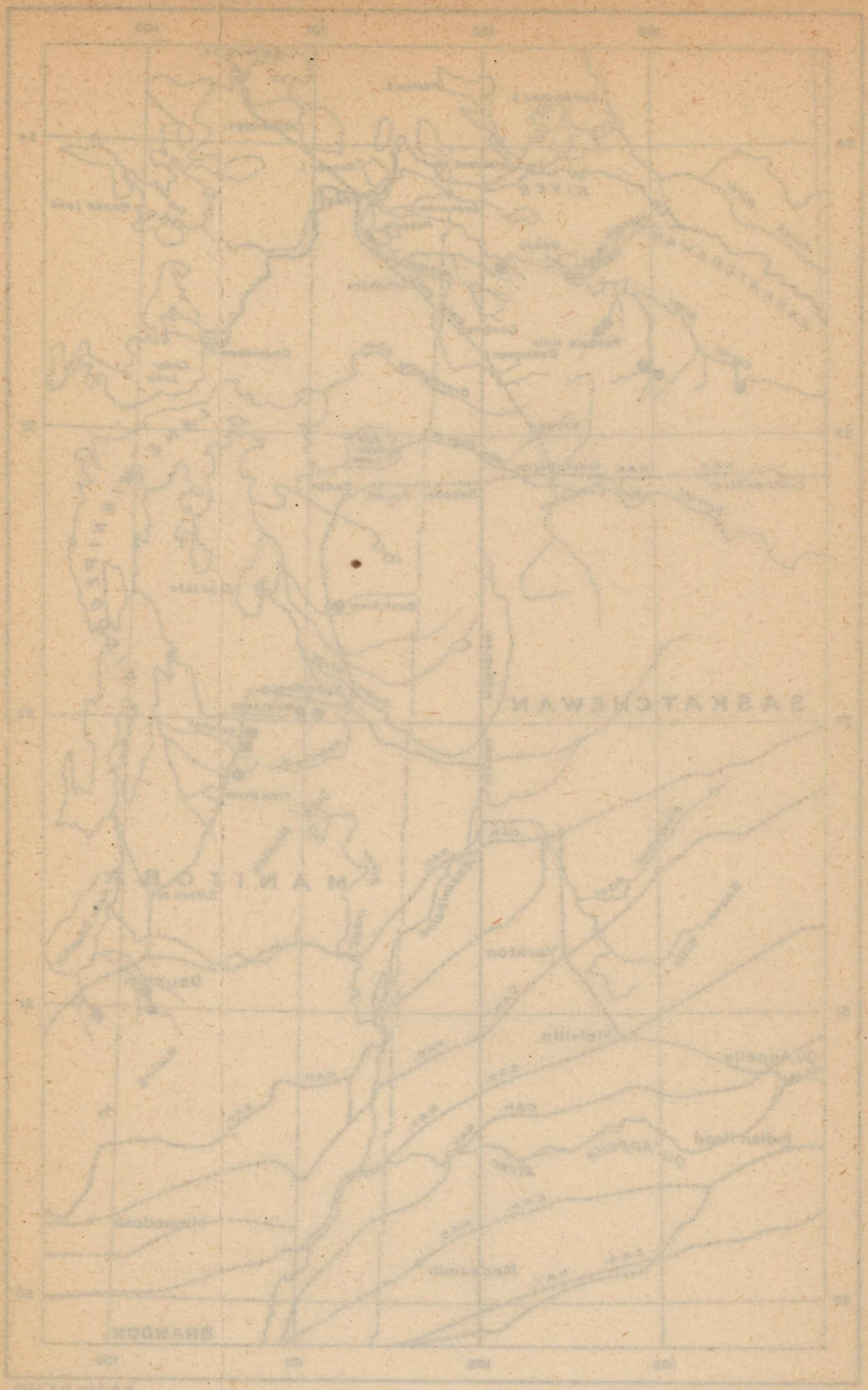
S.C. Ells, C.E. 1921

Map showing position of Outcrops of Cretaceous Shales sampled during year 1921, in Manitoba and Saskatchewan

Scale of Miles

20 10 0 20 40 60 80

FIG. 2.


SASKATCHEWAN
MANITOBA
BRANDON

the drilling tools, in the presence of water, the shale forms an emulsion of a dark green colour. This emulsion gives off a faint odour of petroleum, and, after standing for a time, exhibits a thin film of petroleum on the surface. It is, therefore, not surprising that the presence of free petroleum in wells drilled through the shales has been reported from time to time.

Of Upper Cretaceous shales, the Niobrara formation is the most highly fossiliferous, hence it is within this horizon that bands of oil shales might have been anticipated. Fossils observed in this formation comprise a large number of foraminifera, among which Globigerina cretacea is conspicuous, and an abundance of shells of a large Inoceramus, together with the following species:—[1]

Serpula semicoalita (Whiteaves. N. sp.).
Ostrea congesta (Conrad).
Anomia obliqua (Meek and Hayden).
Inoceramus problematicus (Schlotheim).
Belemnitella Manitobensis (Whiteaves. N. sp.).
Loricula Canadensis (Whiteaves. N. sp.).
Ptychodus parvulus (Whiteaves. N. sp.).
Lamna Manitobensis (Whiteaves. N. sp. loose).
Enchodus Shumardii (Leidy).
Cladocyclus occidentalis (Leidy).

ORIGIN OF SHALE

Oil shales of New Brunswick and Nova Scotia were deposited under different conditions from those of Manitoba and Saskatchewan. In the former provinces, material from which the shale beds were derived was originally deposited in the form of fine clays on the bottom of swamps and lagoons. If the theory of Steuart[2] is accepted, there was associated with these sediments vegetable matter converted into a pulp, as a result of maceration and microbe action in water, together with richer materials of various kinds, such as spores, and a proportion of animal matter. We may, therefore, conclude that the origin of the petroleum in the shales may be traced to fermentation and decomposition of organic matter, through microbe action. On the other hand, the shales of Manitoba and Saskatchewan were laid down in a muddy sea, and were, therefore, not subjected to the various influences that affected the shales of the Maritime Provinces.

PHYSICAL CHARACTERISTICS OF SHALES IN THE PROVINCES OF MANITOBA AND SASKATCHEWAN

It is evident that great areas of Cretaceous shales, with their notable content of organic matter, have been subjected to very slight disturbance. Had the shales been affected by folding or by compression, as a result of great earth movements, distillation of organic remains, due to heat and pressure, would doubtless have resulted in the formation of gas and petroleum.

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[1] Geol. Surv. Report on North-Western Can., Manitoba, J. B. Tyrrell, 1892.
[2] Steuart, D. R., The Chemistry of Oil Shales. Oil Shales of the Lothians, Geol. Surv., Scotland, 1912.

Attempts to ignite fine splinters of the various shales examined, by means of a match, failed in nearly every instance. Niobrara shales, when partially dried, will, however, frequently ignite when placed on an open wood fire. Pierre shales examined ignite but rarely, and then with difficulty. Benton shales showed no evidence of being combustible. Sampling was, therefore, confined largely to Niobrara shales.

In appearance and physical characteristics, as well as in the conditions under which they were deposited, these upper Cretaceous shales are thus in marked contrast to the oil shales of Nova Scotia and New Brunswick. In these provinces, two general types of shale are recognized, viz., plain and curly. A sub-variety, usually referred to as paper shale, is apparently a weathered form of plain oil shale. It is thin-bedded, or papyraceous, separating readily into thin, flexible sheets of considerable surface dimensions. Plain oil shales are usually flat surfaced, showing more or less well defined lamination. On the other hand, curly shale usually occurs as massive bands, somewhat curled and contorted. It is very tough, breaks with a conchoidal fracture, and is, usually very rich in hydrocarbons.

As a rule, oil shales of Nova Scotia and New Brunswick are free from grit, and, although readily cut with a sharp knife, do not yield thin, flexible shavings as do the Scotch shales. They are strongly resistant to weathering, and fragments have remained, for many years, exposed to action of atmospheric agencies, with but little loss of their hydrocarbon content. When placed in a grate, fragments may be readily ignited, giving off an intense heat, and continuing to burn with a long yellow flame for periods of from one to three hours. Small splinters of the shale may usually be ignited by means of a match.

Oil shales of New Brunswick, Nova Scotia, and Scotland, do not contain hydrocarbons which are liquid at atmospheric temperature. Numerous wells have been drilled through the shales by various companies, but, although free petroleum has been encountered in interbedded sands and sandstones, practically none has been met with in the shales themselves.

WEATHERING OF SHALES

Weathering of the Cretaceous shales presents an interesting study. As a rule, the hydrocarbon content of shales examined does not prevent rapid weathering, while, on the other hand, the shales contain within themselves a variety of potent disruptive agencies. Apart from the high percentage of water present, this disintegration, due to the formation of secondary minerals, as iron and calcium sulphates, hydrated iron oxides, etc., and circulating solutions containing acids and sulphates, is very marked.

ASSOCIATED MINERALS

Incidental to the examination of the Cretaceous shales, the presence of certain associated minerals was noted. None of these minerals were observed in sufficient quantity to be of present economic importance, but, pending a complete chemical analysis of the shales, their somewhat complex geochemistry may be briefly alluded to.

These associated minerals fall into two classes, viz., those which, being deep seated, have not been subjected to oxidizing influences, and others, which have been deposited at, or near the surface, in the presence of circulating waters, and under the influence of other oxidizing agencies. In

addition to bentonite, the principal deep seated minerals comprise pyrite, siderite, and glauconite. Owing to the length of time during which they have been forming, these minerals occur in considerable quantity and exhibit comparative purity. Surface deposits comprise marcasite, iron sulphate, limonite, gypsum (in the forms of selenite and satinspar), native sulphur, and small amounts of iron oxide. Pyrite, in the form of small cubes, sometimes occurs in narrow lenses, but is more often disseminated and invisible to the naked eye. Siderite is most abundant in the Pierre series, at times forming beds a few inches thick. Bentonite occurs as uniform, sharply defined beds, having a maximum observed thickness of 24 inches. Pyrite and marcasite, in the presence of oxidation agents, give rise to the formation of small quantities of sulphurous and sulphuric acids. It may be possible that these acids, circulating through the shales, form soluble sulphates which are subsequently leached out, leaving bentonite as a residual product. To the oxidation of iron sulphides may also be attributed the nauseating sulphurous odour which, at times, emanates from shale banks, and has frequently been construed as indicating the presence of natural gas.

Absorption of oxygen, assisted by oxidation of iron sulphides, is responsible for slow spontaneous combustion in progress in many of the banks of shale. At one locality a column of thin blue smoke was observed rising from a smouldering bank, the talus at the foot of the bank being cemented together by various sulphates. Moreover, traces of free petroleum, released as a result of recognized chemical reactions, as well as small amounts of colloidal iron hydroxide, frequently accumulate to form a scum, which, at times, has been mistaken for a true petroleum seepage.

SAMPLING AND ANALYSES

In securing samples overburden was removed and trenches cut from top to bottom of exposed sections. The depth at which samples were taken varied from four to eight feet from the face, but it is considered that in each instance unaltered shale was procured. A number of samples were tested in the field, and satisfactory results were obtained by the use of a retorting apparatus[1] somewhat similar to that recommended by the United States Bureau of Mines. Only the petroleum content was determined. In the case of certain other samples, subsequently tested in the laboratory, the yield of ammonium sulphate was also determined. Twenty-two representative samples were procured on the northern branch of the Pas river, Man river, Cracking river, Papikwan river, Jordan river, and Carrot river, in the province of Saskatchewan, and nineteen samples from Steeprock river, Birch river, Favel river (east and west branches), Sinclair river, North Duck river, Sclater river, North Pine river, Vermilion river, and Ochre river, in the province of Manitoba.

Certain of these samples showed merely a trace of hydrocarbons, hence reference to them is omitted in the following summary.[2]

[1] Reports of Investigations, U.S. Bureau of Mines, Serial No. 2229.
[2] Determinations by Chemical Laboratories, Mines Branch, Department of Mines.

Analyses

Locality	Imperial gals. crude petroleum per ton[1]	Spr. gr. of crude petroleum at 60°F.	Imperial gals. water per ton	Pounds ammonium sulphate per ton
Tee river, Sask., (2 samples) Tp. 53, R. 1, W. of 2nd	3·1— 5·3	0·972—0·984	12·5—16·0	.....
Man river, Sask., (8 samples), Tp. 50, R. 5, W. of 2nd	5·0— 9·4	0·942—0·964	.....	3·1—8·7
Cracking river, Sask., (6 samples) Tp. 49, R. 7, W. of 2nd	7·0—12·8	0·949—0·970	11·6—28·0	2·6—4·5
Jordan river, Sask., (float), Sec. 6, Tp. 48, R. 10, W. of 2nd	10·9	0·964	.....	3·0
Carrot river, Sask., (2 samples), Sec. 25, Tp. 48, R. 11, W. of 2nd	4·9— 6·0	0·958—0·959	.....	1·7—2·8
Birch river, Man., Sec. 31, Tp. 39, R. 26, W. of 1st	0	.....	42·7	.....
Favel river, Man., Sec. 30, Tp. 35, R. 25, W. of 1st	6·2	0·972	12·1	.....
Sec. 26, Tp. 35, R. 26, W. of 1st	6·8	0·984	7·0	.....
" " " " "	5·9	0·965	15·2	.....
Sclater river, Man., (2 samples), Sec. 15, Tp. 34, R. 23, W. of 1st	4·8— 7·5	0·966—0·968	9·2—18·7	.....
Pine river, Man., Sec. 6, Tp. 33, R. 22, W. of 1st	3·3	0·969	4·5	.....
Vermilion river, Man. (2 samples), Sec. 12, Tp. 24, R. 20, W. of 1st	1·1— 5·1	0·952	8·1—22·0	.....
Ochre river, Man., (2 samples), Sec. 29, Tp. 22, R. 17, W. of 1st	4·0— 5·3	0·955	14·6—15·2	.....

[1]Calculations based on ton of 2,000 pounds.

The sulphur content of the crude petroleum recovered from six representative samples of shale varied from 5·3 per cent to 7·7 per cent, with an average for the six samples of 6·5 per cent. A sample of crude petroleum distilled from shale secured on Ochre river, Man., (Sec. 29, Tp. 22, R. 17, W. of 1st) was fractionated with the following result:—

Distillation range	Barom. 766·0 mm.
1st drop	79° C.
10 p.c.	149° C.
20 "	170° C.
30 "	221° C.
40 "	254° C.
50 "	281° C.
60 "	301° C.
70 "	336° C.
80 "	350° C.
90 "	361° C.
92 " dry point (cracking occurred).	

The gas yield from 13 representative samples of shale was also determined, and varied from 410 to 1,130 cubic feet per ton, with an average yield of 695 cubic feet. Analyses of two samples of gas recovered from shale secured on Man river, Sask., (Tp. 50, R. 5, W. of 2nd) are as follows:—

	Sample No. 1825	Sample No. 1831	
	p.c.	p.c.	
Carbon dioxide	32·6	23·3	
Oxygen	2·1	3·1	
Ethylene	3·7	3·8	
Carbon monoxide	2·4	1·9	
Methane	28·7	30·6	
Hydrogen	9·1	7·5	
Nitrogen	21·4	29·8	
Inflammable gas	43·9	43·8	
Calorific value, gross	383	297	B.Th.U. per cu. ft.
Calorific value, net	346	359	"

CONCLUSION

It is, therefore, obvious that shales examined during the past season in the provinces of Manitoba and Saskatchewan are of little present economic importance as a possible source of petroleum or of ammonium sulphate. Should conditions at any time warrant commercial development, open cut mining could be undertaken in many areas under favourable conditions. Over very considerable areas the shales examined are covered by a comparatively light overburden, consisting chiefly of boulder clays and gravel, which could be readily removed by hydraulicing methods.

VII

OIL SHALES OF CANADA

S. C. Ells

(SUMMARY AND REVIEW OF AVAILABLE INFORMATION)

Occurrences of petroliferous shales (commonly termed oil shales), within the Dominion of Canada, have, for many years, been known to exist. Nevertheless, with the exception of a comparatively brief period, Canada has had to depend largely on foreign sources for its supplies of crude petroleum and petroleum products. In 1859, near Collingwood, Ontario, and again in 1862 near Baltimore, New Brunswick, attempts were made to retort local shales. In each instance these attempts were abandoned owing to the rapidly increasing production of well petroleum from the newly discovered pools of western Ontario and Pennsylvania. Now, after a lapse of nearly sixty years, attention is once more directed to the commercial value of Canadian shales as a possible source of crude petroleum.

Thus it is that only within very recent years, changing economic conditions appeared to warrant detailed study of existing shale bodies in Canada. Consequently, much work still remains to be done, not only in the field but in the laboratory, before the economic importance of such areas can be definitely determined. Meanwhile, such information as is available, may be credited almost wholly to the Geological Survey of Canada. The writer is personally familiar with the oil shales of Nova Scotia, New Brunswick, Gaspe peninsula, northern Ontario, Manitoba, and Saskatchewan. Reference to occurrences outside of these areas is based on information secured indirectly.

In the following summarized references to known or reported deposits of oil shales, occurrences will be grouped under the province or district in which they occur. In discussing shales, which vary widely both in physical characteristics and in hydrocarbon content, the use of the term oil shale is in certain instances provisional, and is necessarily applied in a broad sense.

In referring to the oil shales of Canada, it is not the writer's intention to discuss retorting methods and economic considerations. Fundamental principles relating to the former are well understood, though their application to various types of Canadian shale is as yet undeveloped. Reference to economic considerations would of necessity merely constitute a recapitulation of authoritative statements and conclusions. With these, all who are really interested in the commercial development of oil shales, are already familiar. Where oil shales can be cheaply mined by open cut methods, with reasonably high recovery of petroleum and of ammonium sulphate, competition with well petroleum may be possible at the present time. Under conditions involving the adoption of underground mining methods, the possibility of successful competition is by no means clear.

Province of Nova Scotia

The occurrence of oil shale in the province of Nova Scotia has been known for more than 60 years.[1] During the three years following its discovery, several thousand tons of a particularly rich sub-variety, known as stellarite (stellar coal or oil coal), was mined near Stellarton, in Pictou county. This output was derived chiefly from a rich band immediately underlying the McGregor coal seam; but shipments were also made from McLellan's brook, and from a point just west of Blackwood's mill dam. It is stated[2] that stellarite from the first named locality yielded upwards of 10,000 cubic feet of 36 candle gas, and that selected samples gave a yield equivalent to 170 Imperial gallons (204 U.S. gallons), of crude petroleum per ton. The average petroleum content of a cargo shipped to Boston was 48 Imperial gallons (57·6 U.S. gallons), while the associated shale "bat," gave a yield of 39 Imperial gallons (46·8 U.S. gallons). Part of the above output was shipped to retorting works in Boston and in Portland, and part was mixed with bituminous coals for the enrichment of illuminating gas. Following the discovery of well petroleum in Ontario and in the United States, the mining of stellarite ceased.

Oil shales of Nova Scotia are associated, in part, with strata of Lower Carboniferous age, and in part, with the Coal Measures and Upper Carboniferous rocks. The shales, as a whole, differ in character from those of New Brunswick, and with notable exceptions, as in Pictou county, are carbonaceous rather than bituminous. There is a general absence of the brown shales and hard massive oil bands, so typical of the New Brunswick field, and certain large deposits, which were at one time classed as oil shales, have been found to be too poor in hydrocarbons to warrant further consideration.

[1] Ells, R. W., Can. Dept. of Mines, Mem. 1107, 1910.
Ells, R. W., Jour. Min. Soc. of N.S., Vol. XIV, 1910-11.
[2] Poole, H. S., Trans. N.S. Inst. Sci., Vol. VIII, p. 340, 1893.

Areas of reported oil shales extend from near the mouth of the Avon river to the vicinity of East Bay, Cape Breton. Certain of these will be briefly discussed.

COLCHESTER AND HANTS COUNTIES

Along the south coast of Minas Basin, heavy beds of shales are well exposed, notably at Cheverie, between Split Rock and the village of Noel, and on the west side of the Avon, between Hantsport and Horton Bluff. The complete series consists of quartzose grits, interstratified with thick bands of carbonaceous shales, similar to those of Hallowell Grant, Antigonish county, and East Bay, Cape Breton. Certain of these shales will burn, and have been used as a substitute for coal, while others, full of rootlets, constitute true underclays. To the north of Minas Basin, similar shales extend along the south flank of Cobequid mountains, and are well exposed along Moose river, Harrington river, on the North river near Onslow, and elsewhere. These shales were styled the Horton series by Sir Wm. Dawson, and have been correlated with the Albert series of New Brunswick. As a class they are deficient in hydrocarbons, although further detailed examination may possibly discover associated oil shale bands. With these beds are frequently associated beds of dirty coal, high in ash, and of little commercial value.

PICTOU COUNTY

From present information, deposits of Pictou county comprise the most promising oil shales of the province of Nova Scotia. The principal occurrences are found in that portion of the Pictou bituminous coal field centering about the town of Stellarton.

The field has an area of approximately 20 square miles, and is chiefly occupied by Coal Measures and rocks of Lower Carboniferous age. Geological features of the area have been somewhat fully described[1] in publications issued by the Geological Survey of Canada, and in papers presented before technical and scientific societies.

The field may be divided into two main districts, eastern and western. In the eastern district coal seams occur in two horizons of the Productive Measures. In the upper horizon are five seams, varying in thickness from 3 feet to 8 feet. In the lower horizon six seams vary in thickness from 7 feet to 45 feet. With these latter seams have been correlated the four principal seams of the western district. Interposed between the upper and lower coal horizons are strata consisting chiefly of dark shales, and having an aggregate thickness of approximately 1,600 feet. With these are associated the principal oil shale bands.

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[1] Dawson, J. W., Proc. Geol. Soc., Lon., vol. IV, pp. 272–81, 1843–45.
Dawson, J. W., Quart. Jour. Geol. Soc. Lon., vol. X, pp. 42–47, 1854.
Dawson, J. W., Quart. Jour. Geol. Soc. Lon., vol. XXX, pp. 209–18, 1874.
Dawson, J. W., Acadian Geology, fourth edition, supplement to second edition.
Dawson, J. W., Acadian Geology, fourth edition.
Fletcher, Hugh, Geol. Surv. Can., vol. II, 1886, part P.
Fletcher, Hugh, Geol. Surv. Can., vol. V, 1890–91, part P.
Hartley, E., Geol. Surv. Can., Report of Progress, 1866–69.
Lambe, L. M., Proc. and Trans. Roy. Soc. Can., vol. V, sec. 4, 1911.
Logan, W. E., Geol. Surv. Can., Report of Progress, 1866–69.
Poole, H. S., Proc. and Trans. N.S. Inst. Sci., vol. VIII, 1890–94.
Young, G. A. and J. E. Hyde, Geol. Surv. Can., Guide Book No. 1, 1913.

Major and minor faulting has affected the whole district in a marked degree. A comparison of sections also indicates that coal seams vary widely in thickness from place to place, and that deterioration, due to replacement by shales and sandstones, frequently occurs within comparatively narrow limits. These features are of importance in considering the possible economic importance of associated oil shale bands.

Outcrops of certain of the more important bands of oil shale occur near the town of Stellarton, and along McCulloch's brook, McLellan's brook, Marsh brook (a branch of McLellan's brook), Shale brook, on the south shore of Deacon cove, and near the mouth of Smelt brook. The position of these is indicated on a map[1] of the area issued by the Geological Survey of Canada in 1904. Of these oil shales the sub-variety known as stellarite is particularly worthy of note. The following reference to this mineral is by Sir J. W. Dawson:—[2]

> The material known as stellar coal is, as I have maintained in previous publications, of the nature of an earthy bitumen, and geologically is to be regarded as an underclay or fossil soil, extremely rich in bituminous matter, derived from decayed and comminuted vegetable substance . . . .

Stellarite underlies the McGregor coal seams at Stellarton, and also outcrops near the old Patrick slope in McLellan's brook, and on Marsh brook. The thickness of the various seams shows marked variation, and, at different points, has been estimated as ranging from 1 inch to 8 feet.

The following analyses[3] are based on samples of oil shales, other than stellarite, secured by the writer in 1909 from a number of outcrops in the Pictou field:

Locality	Imperial gals. crude oil per ton	U.S. gals. crude oil per ton	Sp. gr.	Pounds ammonium sulphate per ton
McLellan's brook, (Patrick's slope)	38	45·6	0·892	41
McLellan's brook, (Black's old mill site)	15	18·0	0·889	35
Marsh brook (150 feet above McKay's house)	3	3·6	.....	.....
Marsh brook (300 feet above McKay's house)	1	16·8	0·903	.....
Shale brook (upper end)	4	4·8	0·921	.....
Shale brook (near forks with McLellan's brook)	9	10·8	.....	.....
Woodburn sta. (one mile west of, on C.N. Ry.)	13	15·6	0·902	.....

From the above analyses[3] it will be seen that, as a whole, shales from the localities noted are not of commercial value. Results of analyses of shales (including stellarite), from Stellarton and McLellan's brook, indicate that the seams represented are worthy of detailed investigation.

At the present time, plans are being made by the Oil and Nitrate Products, Limited, to develop certain of the shale deposits referred to above. It is claimed that a very large tonnage of shale can be cheaply mined.

[1] Poole, H. S., geological map of the Pictou coal field, Geol. Surv. Can. 1904.

[2] Dawson, J. W., Acadian Geology, 1868, p. 339.

[3] All analyses quoted in this paper are by the Chemical Division, Mines Branch, Can. Department of Mines, Ottawa. Compilations are based on a ton of 2,000 lbs.

ANTIGONISH COUNTY

Shales of Lower Carboniferous age occupy a basin lying immediately to the north of the town of Antigonish. For the most part the shales are black and carbonaceous, thus bearing a marked similarity to those of Hants county. A large number of samples were secured within this area by the writer in 1909, and results of the laboratory determination of the more promising are as follows:—

Locality	Imp. gals. crude per ton	U.S. gals. crude per ton	Sp. gr. crude petroleum	Pounds Am. sulph. per ton
Sawmill brook No. 1	9·7	11·6	·906	19·5
" " No. 2	8·9	10·7	·893	33·8
" " No. 3	8·9	10·7	.....	30·2
" " No. 4	20·4	24·5	.....	30·2

Certain of the outcrops represented by the above analyses are of considerable thickness and extent. Although the analyses do not indicate the presence of bodies of high grade shales, yet conditions on Sawmill brook appear to warrant further investigation.

CAPE BRETON

Shales, slightly bituminous in character, are exposed near Lake Ainslie and about the west shore of McAdam lake. None of these shales appear to have commercial value.

Province of New Brunswick

Of the oil shales of Canada, the most important are those which occur in the province of New Brunswick. This statement is based on incomplete data available at the present time. The oil shales constitute a part of the Albert shale series, the geological position of which is shown in the following ascending section:—

Palaeozoic	Carboniferous	Millstone Grit (grey and purple tinted shales, sandstones and conglomerates).
		Slight unconformity.
		Lower Carboniferous (gypsum, limestones, marls, shales, sandstones and conglomerates generally red or green in colour).
		Unconformity.
		Albert Series (black, brown and grey oil shales with numerous sandstone beds, grits and conglomerates).
Pre-Carboniferous		Unconformity.
		(Slate, schist, diorite, and granite.)

Although the boundaries of the Albert shale series have not been accurately defined, the series constitutes a belt which may be traced through a distance of some seventy miles, from the Memramcook river on the east, to within a short distance of the village of Hampton. It is evident that the series extends still farther westward, since it may be readily

recognized in the grey and black shales associated with grey and brown sandstones exposed on Kennebecasis island, a few miles north of the city of St. John.

In many places the rocks of the series are concealed by drift and by unconformable Lower Carboniferous sediments, but as a result of folding, the shales reappear at a number of points along the general line of strike. In passing westward, the bituminous character of the shales becomes less marked.

The northern boundary of the Albert shale series has not been accurately defined. To the west of Albert Mines the shales dip away from the Caledonia mountains at various angles. Some six miles to the north, in the Stony Creek oil and gas field, the series approaches the surface in the form of a structural terrace. North of the Stony Creek field the structure is obscure, but it appears that the steeply inclined shales of Indian mountain, 7 miles north of Moncton, also belong to the Albert shale series, and represent the northern denuded limb of a broad syncline. Between the three points the Albert series is entirely obscured by Lower and Middle Carboniferous sediments.

The principal areas within which oil shales of commercial value have been recognized lie within the counties of Albert and Westmorland, and their approximate position is indicated on geological and topographical maps issued by the Geological Survey of Canada.[1] The actual areal extent of workable beds of oil shale has not been determined, but outcrops of shale of commercial quality have been recognized near Upper Dorchester, Taylor Village (or Taylorville), St. Joseph's College, Albert Mines, Rosevale P.O., Pleasant Vale, Mapleton, and near Catamount station, and also along Downing creek, Frederick brook, East and West branches Turtle creek, Hayward brook, Prosser brook, and Montgomery brook. It will thus be seen that the area underlaid by oil shale of commercial value is of very considerable extent. At a number of localities, notably at Taylor Village, Albert Mines, Baltimore, Rosevale, and along the east and west branches of Turtle creek, beds of shale have been prospected to some extent by means of test pits and tunnels.[2] As a result of this work, it is evident that, at each of the localities noted, a number of oil shale bands of good quality and of workable thickness occur. Owing, however, chiefly to the effects of faulting and the prevalence of overburden, it is as yet impossible to indicate definitely the number and average thickness of such bands. This is particularly true of the Albert Mines area. In this area there apparently exists a great thickness of so-called paper shales, but

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[1] Geological map of the Lower Carboniferous rocks of Albert and Westmorland counties, New Brunswick, showing the distribution of the Albert shales (1878). Scale, 1 mile to 1 inch.

Map sheet 1. N.E. (Grand Lake Sheet), province of New Brunswick, 1880. (Comprises parts of Sunbury, Queens, Kings, Albert, and Westmorland counties). Scale, 4 miles to 1 inch.

Map Sheet No. 4. N.W. (Cumberland coal field sheet). Province of Nova Scotia and part of New Brunswick (1886). Scale, 4 miles to 1 inch.

Reconnaissance map of parts of Albert and Westmorland counties, New Brunswick (Map 35A). Scale, 1 mile to 1 inch.

Moncton, Westmorland and Albert counties, New Brunswick (Map 63A) 1917. Topography. Scale, about 1 mile to 1 inch.

New Brunswick. Carboniferous areas, and positions of certain shale and clay deposits. (Map 108A). 1913. Scale, 20 miles to 1 inch.

Diagram showing portion of Albert Mines oil shale area, New Brunswick. (Geol. Surv. Can. Summ. Rept. 1913, p. 226).

Albert Mines Oil Shale Area, Albert county, N.B., Geol. Surv. Can. Publication No. 1833.

[2] Can. Dept. Mines. Report No. 1107, pp. 19–21.

owing to complex distortion of the rocks, through faulting and folding, a definite pronouncement regarding possible tonnage commercially available must await further underground exploration.[1]

Apart from surface indications, considerable additional information has been secured through the drilling of numerous deep wells. During the period 1899-1906, a local organization known as the New Brunswick Petroleum Company drilled upwards of 80 wells, chiefly in Westmorland county between the Petitcodiac and Memramcook rivers. The majority of these wells were comparatively shallow, varying in depth up to 1,100 feet, although one at least reached a depth of more than 3,000 feet. In a number of these wells, small showings of petroleum were encountered, but the writer has no definite information regarding the number or character of the oil shale bands passed through.

Since 1909, Maritime Oilfields, Ltd., succeeded by its subsidiary, New Brunswick Oil and Gas Fields, Ltd., has successfully carried on an extensive drilling programme in the Stony Creek oil and gas field, 9 miles south of Moncton, and elsewhere. The wells in the Stony Creek field are located on a structural terrace, and vary in depth from 1,800 to 3,200 feet. Their aggregate shut in capacity has exceeded 100,000,000 cubic feet of gas every 24 hours. The gas is found in a large number of interbedded sandstones, ranging in thickness up to 100 feet, and grouped along indefinite horizons through a vertical range of 2,000 feet.

More recently, drilling to the dip of this gas area has resulted in a moderate production of high quality paraffin base petroleum, and during 1920 the year's production exceeded 6,300 barrels. The greatest thickness of the Albert shale series recorded by any log from the Stony Creek field is approximately 2,800 feet, and in this log the occurrence of some 16 oil shale bands is noted. Of these it is considered that six bands, with an aggregate thickness of 72 feet, are of commercial quality, while the remainder, aggregating in thickness 123 feet, are of somewhat poor quality. Interbedded with these oil shale bands are numerous other sandy, marly and limey shales and thin limestones, which show wide variation in thickness, character, and texture. The total original thickness of the Albert shale series, estimated to be not less than 7,000 feet, would doubtless include a much larger number of oil shale bands of commercial grade. It is considered that the richer bands of oil shale occur in the lower horizons of the series.

Owing to lateral variation, however, it has been found that, even when the horizontal distance between individual wells does not exceed 1,000 feet, serious difficulty exists in correlating individual beds recorded in the logs. This appears to indicate that, in parts of the field at least, rapid changes in thickness and in character of oil shale bands and associated strata may be anticipated. This tendency is in conformity with the habit of associated sandstones and shales of the Carboniferous series of eastern New Brunswick, many of which were laid down in basins of limited extent.

Between 1911 and 1913 other private interests, said to be identified with Sir Wm. Mackenzie, tested the area along the southern margin of the Albert shale basin, by means of diamond drills. It is said that upwards of 14 holes, varying in depth from 700 to 1,500 feet, were drilled in the

[1] Wright, W. J., Geol. Surv. Can., Summ. Rept. 1913 (pp. 223-227).

vicinity of Rosevale P.O., between the east and west branches of Turtle creek, while upwards of 10 other holes were drilled at scattered points. During the period 1919-21, eight other exploratory wells, some of which reached a depth of more than 3,000 feet, have been drilled in other parts of the Carboniferous basin of eastern New Brunswick, by the D'Arcy Exploration Company. The depth of most of these wells exceeds 2,000 feet.

Throughout the geological basin occupied by the Albert shale series notable erosion has taken place. In parts of the basin, several thousand feet of the series have been removed, and subsequently replaced by, unconformable beds of Middle and Lower Carboniferous age. Obviously, under such conditions, only extensive and systematic drilling will indicate the extent to which associated oil shale bands have been affected.

Oil shale bands, associated with the Albert shale series, vary in thickness from a few inches to more than 25 feet. In most instances the shale kindles readily and burns freely. Small splinters, when ignited by a match, frequently support combustion for considerable periods. In general, the oil shales weather through various shades of grey, but the colour of freshly broken surfaces ranges from grey to a typical chocolate brown or brownish black. The shales frequently contain thin laminæ or partings of asphaltic bitumen resembling albertite.

In point of geological age, and in certain other respects, the oil shales of New Brunswick correspond somewhat closely with those of Scotland. In each instance two general types of oil shales are recognized, viz., plain and curly. A sub-variety, usually referred to as paper shale, is apparently a weathered form of plain oil shale. It is thin-bedded, or papyraceous, separating readily into thin, flexible sheets of considerable surface dimensions. In many instances, the surface of the sheets are almost covered with imprints of fossil fish remains. Plain oil shales are usually flat-surfaced, showing more or less well-defined lamination. On the other hand, curly shale usually occurs as massive bands, somewhat curled and contorted. It is very tough, breaks with a conchoidal fracture, and is usually especially rich in hydrocarbons, a condition which may have rendered it more easily crumpled than poorer grades of shale. As a rule, the New Brunswick shales are free from grit, and, although easily cut with a knife, do not yield thin, somewhat flexible shavings as do the Scotch shales.

In view of the similarity which exists between the oil shales of Scotland and those of New Brunswick, each may probably be attributed to a somewhat similar origin. In both countries there is little in the shale which is soluble in benzene, carbon disulphide, ether and other similar solvents, and it would therefore appear that petroleum derived by destructive distillation may be attributed to matter of organic origin[1]. To express this carbonaceous substance, the term kerogen has been suggested.

Apparently, material from which the shale beds were derived was originally deposited in the form of fine clays on the bottom of lagoons and swamps. If the theory of Steuart is accepted, there was associated with these sediments vegetable matter converted into a pulp as a result of maceration and microbe action in water, richer materials of many kinds,

[1] Carne, J. E., "The Kerosene Shales Deposits of New South Wales".
Steuart, D. R., The Chemistry of the Oil Shales. Oil Shales of the Lothians, Geol. Surv. Scotland, 1912.

by an escarpment of Pre-Cambrian rocks. The area is traversed by a number of large streams, notably the Abitibi, Mattagami, and Missinaibi. Along these Palæozoic sections, including strata of Ordovician, Silurian, and Devonian age, are exposed.

The possibility of discovering well petroleum in Palæozoic rocks of the James Bay coastal plain, has recently drawn attention to certain petroliferous shales which outcrop on the Abitibi and Mattagami rivers. These shales, which belong to the Ohio formation, are well exposed at the Long rapids on the Abitibi river, and to a limited extent above the Long rapids on the Mattagami river. Sections exposed along the Abitibi are marked by a series of low folds, the general strike of which is approximately east and west. The maximum thickness of shale observed at any point was approximately 55 feet.

In places the shales are covered by a comparatively light overburden of boulder clay and other unconsolidated materials. The construction of the proposed extension of the Timiskaming and Northern Ontario railway would bring the deposits within reach of rail transportation.

In 1911[1] these shales were examined by the writer, and samples subsequently tested in the laboratory showed a content of petroleum ranging from 7 to 16 Imperial gallons per long ton. The maximum yield of ammonium sulphate was equivalent to 16 pounds per long ton of shale. Partial analyses of three other samples, collected by Dr. M. Y. Williams,[2] indicated a yield of from 3·5 to 12 Imperial gallons (4·2 to 14·4 U.S. gallons) crude petroleum per ton of shale. The calculated yield of ammonium sulphate, based on the nitrogen content, was equivalent to from 18·8 to 38·6 pounds per ton. The amount recoverable in actual commercial practice would be considerably less.

Provinces of Manitoba and Saskatchewan

NOTE.—During the field season of 1921, the writer undertook a reconnaissance with a view of determining the probable economic importance of the Cretaceous shales of the Pasquia hills, Porcupine, Duck and Riding mountains. A complete report embodying the results of this work is in course of preparation.[3]

During recent years, attention has been directed to reported occurrences of oil shales of Cretaceous age in the provinces of Manitoba and Saskatchewan. Forty-one samples of the shales, representative of a wide area, were collected by the writer during the field season of 1921, and were subsequently tested in the laboratory. The maximum yield of crude petroleum from any sample was 10·9 Imperial gallons (13·1 U.S. gallons), while the maximum yield of ammonium sulphate was equivalent to 3 pounds per ton of shale. The specific gravity of the crude petroleum varies from ·944 to ·984. All shales examined carry a high percentage of water, the average content of 15 samples being equivalent to 15·4 Imperial gallons (18·4 U.S. gallons) per ton.

It appears, therefore, that the shales examined in the provinces of Manitoba and Saskatchewan are of little present economic importance as a possible source of petroleum or ammonium sulphate. Should conditions at any time warrant commercial development, open cut mining

[1] Report on James Bay Surveys, 1912, S. C. Ells.
[2] Geol. Surv. Can. Summ. Rept. 1919, Part G.
[3] Mines Branch, Dept. of Mines, Memorandum Series No. 3, 1921.

could be undertaken in many areas under favourable conditions. Over very considerable areas, shales examined are covered by a comparatively light overburden, consisting chiefly of boulder clays and gravel, which could be readily removed by hydraulicing methods.

Province of British Columbia

The occurrence of oil shales has been reported[1] near Harper's camp in the Cariboo district, near Lytton, and along Calder creek, a tributary of Flathead river. Detailed exploration of these occurrences has not been undertaken, but from present information it appears that the maximum yield of crude petroleum from any sample tested does not exceed 7 Imperial gallons per ton.

On Graham island, the most northerly of the Queen Charlotte group, viscous hydrocarbons have a wide distribution throughout sedimentary and intrusive rocks of Cenozoic and Mesozoic age. Semifluid bitumen is seen as thin films along bedding planes and joint surfaces, as veinlets in various fractured rocks, and in amygdules of certain basalts, as at Tian point. This condition was observed by the writer in 1905, and has subsequently been fully described by J. D. Mackenzie in a report[2] dealing with the geology of Graham island.

The presence of traces of bitumen over a wide area on Graham island was, at one time, interpreted as a possible indication of petroleum pools. Mackenzie considers that the bitumen originates in the Maude formation, a series of dark coloured, fine-grained, thinly laminated and highly fossiliferous argillites, of Lower Jurassic age. He considers that the possibility of discovering commercial pools of petroleum, associated with sediments of Graham island, may be regarded as remote, but suggests the possibility that oil shale bands of commercial value may be found associated with the Maude formation.

Provisional District of Mackenzie

From time to time, during the past 30 years, the occurrence of oil shale has been reported[3] along the Mackenzie river between Fort Norman and Fort Good Hope. These shales are associated with rocks of Devonian age, but little information is as yet available regarding their thickness, extent, and value as a possible source of petroleum and various by-products. The outcrops occur some 1,500 miles to the north of the city of Edmonton, the nearest large centre of population, and apart from other considerations, their geographical position thus renders them of little present economic importance.

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[1] Ann. Rept. of Min. of Mines, B.C., 1903, p. 24.
Ann. Rept. of Min. of Mines, B.C., 1904, pp. 23, 24.

[2] Can. Geol. Surv. Mem. 88, 1916.

[3] McConnell, R. G. "An Exploration in the Yukon and Mackenzie basins, N.W.T.", Geol. Surv. Can. Ann. Rept., new series, vol. IV, pt. D, 1888-89 (1890), p. 31.

Kindle, E. M. and Bosworth, T. O., "Oil Bearing Rocks of Lower Mackenzie Valley", Geol. Surv. Can. Summ. Rept., 1920, Pt. B.

Provisional District of Franklin

Among the specimens collected by Captain Bernier during the voyage of the SS. Arctic in 1908, were black oil shales, which were picked up on the beach of Melville island. These specimens are rich in hydrocarbons, kindling readily when ignited by a match. They are apparently of the same class and probably of the same horizon as the oil shales of New Brunswick, Newfoundland, and Bear island, Spitzbergen.

The results of a sample determined in the laboratory of the Mines Branch showed 64 pounds of ammonium sulphate and 140 Imperial gallons of crude oil per ton of shale.

CONCLUSION

In referring to possible commercial development of oil shales in North America, many writers have cited the admitted success which, until recent years, has attended the Scotch oil shale industry.[1] But new and well recognized factors are already reflected in declining output throughout at the Scotch oil shale works. Among these factors may be mentioned higher operating costs, increasing production of well petroleum from new and from established fields, the potential importance of recognized, but as yet unproductive oil bearing areas, and an increasing production—accompanied by declining market values—of ammonium sulphate derived from sources other than oil shale. Certain of these factors may be of temporary significance only.

Certain of the oil shale deposits of Canada warrant careful investigation. These deposits represent a large aggregate tonnage of valuable shale, and there is good reason to suppose that, under reasonably favourable conditions, they will eventually prove of sufficient merit to constitute the basis of a large and lucrative industry.

Commercial development of these deposits will be determined by the question of supply and demand, for it may be assumed that efficient retorts, adapted to the treatment of various types of shale, will be developed. Apparently mining charges will constitute the most important single factor in production costs, and for the immediate present, will adversly affect development. Exceptional instances may exist where shales carry reasonably high values, and where conditions affecting mining operations are unusually favourable.

Should conditions not appear favourable to commercial development in the immediate future, certain of the oil shale deposits of Canada should, nevertheless, be regarded as constituting potentially a valuable national asset.

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[1]Beilby, G. "Thirty Years of Progress in the Shale Oil Industry." Jour. Soc. Chem. Ind., Vol. XVI., pp. 876-886, 1897.
Caldwell, W., "The Working of Oil Shale at Pumpherston." Trans. Inst. Min. Engr. Vol. XXXVI, pp. 581-589, 1909.
Craig, Perkin and others, "The Scotch Oil Shale Industry," 1918.
Ells, R. W., "Bituminous Shales of Nova Scotia and New Brunswick, with notes on the Oil Shales of Scotland." Summ. Rep. Geol. Surv. Can. for 1908 and 1909.
Ells, R. W., "Joint Report on the Bituminous or Oil Shales of New Brunswick and Nova Scotia, also on the Oil Shale Industry of Scotland." Bulletins 55 and 1107, Can. Dept. of Mines, 1909.

VIII

CHEMICAL PRODUCTS FROM NATURAL GAS

R. T. Elworthy

Canada has large resources of natural gas, second only in importance to those of the United States. The chief fields are in western Ontario and Alberta, and the annual production in each province is between ten and fifteen thousand million cubic feet.

The supplies of gas from the Ontario fields are, however, decreasing. The old wells are falling off in output and the high cost of drilling new wells and small return upon capital outlay have retarded the development of new sources. The proper use of natural gas in Ontario is undoubtedly for domestic fuel, and the activities of the provincial authorities have chiefly been in economic enquiries and in securing the greatest efficiency of the combustion apparatus. There is great room for improvement in the efficient use and combustion of natural gas. People regard it as a never-failing gift of nature and use it in a correspondingly spendthrift manner.

The condition of affairs in the west is somewhat different. Although certain fields in the southern part of Alberta, such as the Bow Island field, are showing signs of exhaustion, new borings give indications of large resources. In the Athabaska region great quantities of gas have been struck. During the next decade, with the increased exploratory drilling for oil, many gas fields are bound to be opened up and millions of cubic feet of gas will be allowed to waste.

These supplies of gas are situated too far from centres of population to be used as fuel. What can be done with them?

In a recent and most suggestive paper[1] before the Institute of Petroleum Technologists on oil field wastes, Beeby Thompson states his opinion that the greatest waste is sustained in hydrocarbon gases. He says:—

> Responsible Washington Officials have volunteered the information that in a few years prior to 1912 not less than 425,000,000,000 cubic feet of gas were allowed to discharge unheeded to waste in the mid-continent oil fields alone, equal to about 9,600,000 tons of oil, or say 14,000,000 tons of good quality coal, whilst gasoline to the value of millions sterling was simultaneously lost. In 1913 a single well in the Cushing oil field gave 1,500,000,000 cubic feet of gas, equal to 34,000 tons of oil, before being shut in, and four others were discharging 126,000,000 cubic feet daily into the atmosphere, equal to about 2,800 tons of coal.

Beeby Thompson suggests the following uses for waste gas:—

(*a*) Transmission in pipe lines to industrial centres.
(*b*) Development of local industries like glass-making, lime-burning, cement works, irrigation.
(*c*) Generation of electric power on a large scale.
(*d*) Production of carbon black.
(*e*) Fixation of nitrogen from the air.
(*f*) Isolation of special constituents.

[1] Jour. Inst. of Petroleum Technologists, Vol. 8, pp. 1–49, Jan. 1922.

Such uses are certainly possibilities in many fields in the United States, and they are rapidly being put to application. The situation is different in Canada where the gas supplies of this nature are too inaccessible to be utilized along these lines.

The solution lies in using them for the manufacture of commercially important chemical products by processes requiring simple plant and little labour, utilizing the gas both as the raw material and for the production of the necessary heat and electric power.

The object of this paper is to point out some of the ways in which natural gas can be used as a basis for the manufacture of such chemical products.

The Mines Branch has had this subject under consideration for some time, and although no striking results have been obtained, much information has been collected which it is hoped may be of value to the natural gas industry.

THE CHARACTER OF CANADIAN NATURAL GAS

Although the natural gas fields have been given considerable attention from the geological point of view[1] very little information exists, especially

Analyses of typical Canadian Natural Gases

Source	Estimated flow cu. ft. per day	Methane	Ethane	Oxygen	Carbon dioxide	Nitrogen	Helium
1. Stony Creek, Moncton, N.B.	10,000,000 (1913)	80·0	7·2	.....	.....	12·8	0·06
2. Simcoe, Ont., Dom. Natural Gas Co.	.....	80·3	7·6	.....	0·3	11·8	.....
3. Askew well, Tilbury tp., Kent co.	2,000,000 (1914)	84·4	10·8	(H_2S 0·3)	.....	4·5	0·18
4. Fairbanks well, Enniskillen tp. Lambton co., Ont.	15,000,000 (1914)	68·3	15·9	.....	.....	15·8	0·15
5. Welland No. 2, Bertie tp., Ont.	1,000,000 (1914)	80·0	12·9	0·2	.....	7·1	.....
6. Medicine Hat city gas	.....	90·0	3·0	0·1	0·8	5·0	0·11
7. Bow island, Barnwell well No. 25	4,000,000 (1919)	88·3	3·7	.....	.....	7·0	0·36
8. Sheep river, Calgary, Petroleum Products well No. 1 and 2	5,000,000 (1919)	72·0	16·7	.....	.....	11·3	.....
9. Viking, Alberta.	3,000,000 (1918)	94·0	.....	0·4	0·6	5·0	0·05
10. Pelican rapids, Athabaska river	870,000 (1916)	97·0	.....	.....	1·2	1·8	0·002
11. Kamloops, B.C.	.....	96·8	.....	.....	0·3	2·9	.....
12. Port Haney, B. C.	.....	22·5	.....	.. 7·5	1·2	68·3	0·013

[1] Petroleum and Natural Gas Resources of Canada, Vols. I and II, F. C. Clapp and others, Mines Branch Report No. 291, 1914.

The Oil and Gas Fields of Ontario and Quebec, Wyatt Malcolm, Memoir 81, Geological Survey, 1915.

Investigations in the Gas and Oil Fields of Alberta, Saskatchewan and Manitoba, D. B. Dowling, S. E. Slipper and F. McLean. Memoir 116, Geological Survey, 1919.

Authority—
Nos. 1, 2, 8, 10 and 11 Mines Branch Laboratories
Nos. 3, 4, 5 Messrs. Ellis, Bain and Ardagh
Nos. 6, 7, 9, 12 Mines Branch Bull. No. 31

in the case of the gases in western Canada, concerning the character and nature of the constituents of the chief Canadian gas resources. Table I gives analyses of representative gases.

The chief Ontario gases have been analysed by Messrs. Ellis, Ardagh and Bain, and well described by G. R. Mickle.[1] Their analyses show that the gases are dry gases and usually contain about 80 per cent methane and 10 to 12 per cent ethane. The western gases, as far as the few analyses indicate, seem to be higher in methane, mostly about 90 per cent, though this is probably due to the fact that the analyses are chiefly commercial, with all the hydrocarbons reported as methane, lacking the refinements of the modern gas analysis. There is undoubtedly need for a careful investigation of the western gases.

CHEMICAL PRODUCTS FROM NATURAL GAS

One of the chief developments in chemical technology in recent years has been the number of important chemical processes based on gas reactions. As example may be cited the Haber process for the synthesis of ammonia, ammonia oxidation, the great advances in the production of gaseous fuels, the Shawinigan production of acetone and acetic acid from acetylene, ethyl alcohol from coke oven gas in England, isopropyl alcohol from oil gases, and helium production in Canada and the United States.

The future will see many more advances in this line, especially in the field of hydrocarbon chemistry.

The following, in order of their importance and value, are now being produced on a large scale in the United States.

Gasoline

According to the latest U.S. Geological Survey statistics, of the 5,133,000,000 gallons of gasoline produced in 1920, 383,000,000 gallons, or over seven per cent of the total, were separated from natural gas and casing head gas. In 1919 the amount was about 13 per cent of the production.

The chief methods, fully described in the recent literature,[2] are (1) compression and condensation (2) absorption in mineral oil or by activated charcoal. The charcoal absorption method, developed by Mr. G. Burrell,[3] gives excellent results with gases containing as little as one pint of gasoline per thousand cubic feet, and would undoubtedly be applicable to many Canadian gases.

Dr. D. B. Dowling[4] carried out some tests of the gasoline content of certain gases from the Bow river, Medicine Hat and Viking fields using an oil absorption method. The Bow island gases contained from 1 to 3 pints gasoline hydrocarbons per 1,000 cu. ft. gas. Tests on gases from the other

[1] Ontario Bureau of Mines, 23rd Annual Report, 1914, pp. 237–273.
[2] R. P. Anderson, Jour. Ind. and Eng. Chem., 1920, 12, 547.
[3] Chem. and Met. Eng. 24, 156–160, 1921.
[4] Geol. Sur. Summary Rep. 1918, pt. C, p. 17.

fields gave negligible results. More work of this character should be done. Only one natural gas gasoline plant is in operation in Canada at the present time, at the Dingman well at Okotoks, near Calgary.

Carbon Black

Carbon black, made by burning natural gas with an insufficient supply of air is a very important product. In 1921 fifty-eight million pounds, valued at over $5,000,000, were obtained from 49,000,000,000 cubic feet of gas in the United States. The process is very inefficient, as only about 1 to 1·2 lbs. of black is produced from 1,000 cubic feet of gas, compared with a theoretical yield of about 40 (about 2·5 per cent recovery). Although investigations have been carried out with a view to increasing the efficiency, thus far it has only been obtained by a corresponding sacrifice of quality in the product. Here is an opportunity for the physical chemist.

The chief value of this form of carbon lies in its high absorptive property, and it is estimated[1] that the United States product is used in the industries in the following way:—

35 per cent for printer's ink.
40 per cent as a filler in automobile tyres and rubber goods.
10 per cent miscellaneous.
15 per cent exported (to Canada?)

During the first three months of 1921, 380,715 lbs., valued at $59,053 were imported into Canada, duty free.

The miscellaneous uses include its incorporation in almost every article that is black, such as paints, enamels, stove and shoe polishes, phonograph records, buttons, carbon paper, typewriter ribbons, celluloid, cement colours, drawing and marking inks.

No carbon black is made in Canada at the present time, though several inquiries have been received by the Mines Branch as to suitable locations for carbon black plants. Many of the western gases would probably be satisfactory, but so few complete analyses of these have been made that the proportions of ethane and higher hydrocarbons are not known. The most suitable gases for carbon black manufacture are high in these constituents.

One field which has been considered is at Pelican, on the banks of the Athabaska river, Alta. One well drilled by the Geological Survey in 1897 has yielded ever since a considerable flow of gas, and other wells drilled in more recent years have also afforded proof of the existence of a large reservoir of gas. An analysis of the gas from the government well, made in 1916 by Mr. E. Stansfield, gave:—

Carbon dioxide	1·2 per cent
Methane	97·0 per cent
Nitrogen	1·8 per cent

assuming that the oxygen found in the sample was due to air contamination. Together with gasoline extraction, carbon black manufacture is undoubtedly one of the best uses for waste western gases at the present time.

[1] U.S. Geol. Sur. Mineral Resources of the United States, 1920, Pt. 11, pp. 145–149. Carbon black produced from natural gas in 1920. E. G. Sievers.

Helium

Canada is the only place in the British Empire where helium bearing natural gases are found, and even these contain a lower percentage than the best American sources. The Bow island gas, from which several thousand cubic feet of helium was separated in 1920, due to the efforts of Professor J. C. McLennan, carries about 0·36 per cent. The gas used at the United States plant at Fort Worth, in Texas, from which about 2,300,000 cubic feet of pure helium has been obtained, runs as high as 0·9 per cent. In this connection United States Geological Survey professional paper 121, "Helium bearing natural gas," by A. Sherburne Rogers, is a most valuable and complete study of the subject. The Canadian sources are described in Bulletin No. 31, 1920, Mines Branch, Department of Mines.

Although at the present time there is no market for helium, it would seem a wise policy to investigate every potential source of supply of this gas and to develop the most efficient methods of separation, for, undoubtedly, the scientist will find very valuable uses for helium, and great developments in airship transportation will take place when the European countries recover from the paralysing effects of the great war.

Halogen Derivatives

Natural gas chlorination is a well established chemical process, the chief products being methyl chloride, chloroform, and carbon tetrachloride. Several Canadian patents have been taken out, but none of them have been established on a manufacturing scale in this country.

Some experiments have been carried out in the Mines Branch on the iodation of natural gas. The heat of formation of methyl iodide is much lower than the corresponding heat of formation of methyl chloride, and the reaction should be more controllable so that the monohalide could be made the primary product.

The best arrangement of apparatus was found to be: natural gas→flow meter→preheater→iodine vapourizer→heated catalyst tube→condensing coil→wash bottles→gasometer.

Activated charcoal was used as the catalyst in the first series of experiments. In others the activated charcoal was impregnated with (1) silver, (2) platinum, (3) antimony. The effect of various catalyst temperatures and flow rates was studied. In most experiments small quantities of methyl iodide were formed, but the best yield, at 200° C. with activated charcoal, was only about 4 per cent of the theoretical.

Oxidation Products

Processes using natural gas as a source of formaldehyde and methanol have been the subject of several patents in the United States and Germany, and this problem has been the subject of an investigation in the Mines Branch chemical laboratories for some time. The chief methods examined have been oxidation by passing mixtures of natural gas and oxygen or air over catalysts, oxidation by means of ozone, oxidation by reaction with carbon dioxide, and oxidation in solvents.

The chief catalysts used in the first method were: magnetite, ferric oxide, copper oxide, silver oxide, platinum, thorium oxide, borosilicate glass, uranium oxide and vanadium oxide.

The effect of changing the various factors such as temperature of catalyst, flow rate and ratio of methane to oxygen was studied. The data in the following table were obtained using copper oxide as catalyst and are representative of the results found with the other catalysts. The results were all poor.

CATALYST. Copper oxide on asbestos.

Temp. Catalyst °C.	Ratio CH_4, O_2	Flow rate mixture litres per minute	Space velocity L. per L. Catalyst per hour	Gas analyses						
				Initial gas			Final gas			
				CH_4	O_2	CO_2	CH_4	O_2	CO_2	CO
410	3 : 1	0·10	187	78·5	20·8	0·2	.....	1·8	12·0	.....
410	5 : 1	0·10	210	78·5	14·4	0·4	.....	1·8	6·2	.....
500	3 : 1	0·10	187	74·6	25·1	0·3	.....	4·5	1·3	.....
500	3 : 1	0·10	187	74·2	24·7	0·7	81·6	1·4	16·8	0·2
500	2 : 1	0·10	165	65·0	30·3	0·6	.....	4·2	17·6	0·2
500	5 : 1	0·10	210	78·4	17·4	0·6	.....	2·3	16·3	.....
500	9 : 1	0·10	225	85·2	9·4	0·6	.....	1·4	2·8	.....

In the second method no action occurred when natural gas was mixed with oxygen containing 3 per cent ozone. The gas and oxygen were then mixed before passing through the ozoniser and the mixture subjected to the silent discharge.

A gummy liquid was formed consisting chiefly of polymerised aldehyde, though methanol, formic acid, and formaldehyde were also detected. The apparatus used was not very satisfactory and several explosions occurred. Further experiments with faster flow rates and a better type of ozoniser should be carried out. Only traces of formaldehyde were obtained in the third method.

The results obtained thus far have not been very encouraging, agreeing with the experience of investigators working on the same problem in the United States. Formaldehyde is usually formed by either method, but the yields are very low and far too small for the establishment of a commercial process. Methanol is seldom detected in the products of reaction, though it is a theoretically possible product. Formaldehyde is rapidly becoming a very important and widely used chemical, and a cheap process of manufacture from natural gas would be of great value.

Some work has been done in the Mines Branch laboratories on the hydrogenation of formaldehyde. According to Sabatier and Senderens[1] formaldehyde is quantitatively converted to methanol when passed with hydrogen over nickel at 90° C.

A number of experiments were therefore carried out in which formaldehyde was formed by heating paraformaldehyde in a current of hydrogen, and the mixture of gases passed over nickel catalyst prepared in the usual way by reducing pure nickel nitrate, heated to 300° C. in a current of hydrogen.

[1] Comp. Rend. 137–302.

The gases issuing from the catalyst chamber were scrubbed by passage through a series of wash bottles containing water, and the methanol and unchanged formaldehyde were estimated. The following table gives some of the data obtained:—

Exp. No.	Catalyst	Catalyst temp. °C	Rate of flow hydrogen c.c. per min.	Methanol % of theoretical formed	Total formaldehyde recovered %
VI A	nickel	150	200	15·7	31·9
X A	"	125	300	13·5	43·5
XII A	"	100	300	23·7	61·4
III C	"	150	200	19·0	64·0
I B	platinized asbestos	100	300	23·5	53·2
II B	"	190	300	26·8	54·3
XI B	cobalt	100	200	35·2	70·0
XII B	"	200	200	34·0	54·5
IX B	glass wool	100	300	15·5	87·6

It will be observed that the reaction was far from complete, under the conditions employed. There was also a loss of formaldehyde, which has not yet been quite satisfactorily accounted for.

The most probable explanation is that an acetal or an aldol condensation takes place and experiments now in progress seem to confirm this.

Hydrocarbons

With the increasing development of the synthesis of complex organic substances, such as rubber and camphor, the higher paraffin hydrocarbons, e.g. butane and pentane, will become important, as these substances can be readily converted by chlorination and reduction into higher unsaturated compounds such as isoprene and butadiene, the starting point for rubber snythesis. By hydrolysis the chlorides can be made into various alcohols and thence by oxidation into complex aldehydic and acidic substances.

Butane and pentane occur in some natural gases and casing head gases, and might be isolated. These compounds have never been reported to occur in any Canadian gas, probably because no examination has been made for them. It has been recently reported that a process has been established in Germany for the manufacture of liquid hydrocarbons from methane or natural gas on the lines of the Haber process.

Ethylene

Several suggestions have been made that ethylene might be made from methane by passage over carbon mixed with catalysts.

Ethylene is a gas which is rapidly growing in importance, and methods for its separation from the gases given off in coke oven[1], oil cracking and oil shale retorting plants are being developed, with the subsequent production of ethyl alcohol for fuel purposes.

[1] Jour. Soc. Chem. Ind. 1920, 94, 593A.

Experiments on the formation of ethylene by the passage of natural gas over mixture of heated copper oxide and carbon have been conducted in our laboratories. The greatest percentage of ethylene found under the conditions employed has been 3 per cent. Even this amount is greater than would be expected according to the studies of V. B. Lewes and of Profs. W. A. Bone[1] and H. F. Coward on the decomposition of the simpler hydrocarbons by heat.

Other possible products from ethylene besides alcohol are the chlorinated derivatives and from them by hydrolysis, glycols. The polymerization of ethylene yields higher hydrocarbons of commercial value, though the reactions have been given little attention from an industrial view point.

CONCLUSIONS

This list of products, some of them not yet produced commercially, shows the possibilities of natural gas as a raw material for chemical products of great value, and is a sufficient argument for the expenditure of time and money for the conservation and development of this natural resource. The Mines Branch laboratories are well equipped to carry out much of this work, though the successful solution of these problems will be obtained only by the widest exchange of opinions and by co-operation with the natural gas and chemical industry.

[1] Jour. Chem. Soc. Trans. 93, 1197, 1908.

CHEMICAL DIVISION

F. G. Wait

Chief of Division

During the year 1921, in addition to the usual analytical work, two special lines of investigation were pursued in the chemical laboratory of the Mines Branch, Sussex Street, Ottawa.

I. To discover if it be possible to prepare in commercial quantities methanol (methyl alcohol) and formaldehyde from the natural gas which occurs in several parts of Canada, particularly in Alberta and Saskatchewan.

II. A study of the properties of the colloidal clay known as bentonite, and its possible application in the industries.

The work in both these investigations is still incomplete, but it is realised that sufficient progress has been made to warrant publication of the important results already attained.

I

THE POSSIBILITY OF PRODUCING METHANOL (METHYL ALCOHOL) AND FORMALDEHYDE FROM NATURAL GAS

R. T. Elworthy

During the last three or four years, owing to the efforts made by Great Britain and the United States to establish dye and other organic chemical industries, methanol and formaldehyde have rapidly grown in importance, and increased attention has been paid to their production.

The only source of methanol is from the wood distillation industry; formaldehyde is, of course, manufactured from it.

Methanol, however, is one of the minor products in the distillation of wood, and the supply—until the recent depression in trade—has not kept pace with the demand. When conditions improve it is anticipated that increasing quantities of these essential chemicals will be required. The wood distillation industry will not be able to meet this demand, hence other methods of production must be found. One possible source is by chemical processes, using natural gas as raw material.

THE USES OF METHANOL AND FORMALDEHYDE

Methanol

The chief use of methanol is, probably, in the manufacture of formaldehyde, in which process a very pure grade of alcohol is required. The operation consists in passing a mixture of air and alcohol vapour over hot copper gauze, the success of the reaction being dependent on careful control of the temperature. The resulting formaldehyde is absorbed in water, the solution concentrated, and usually sold in the form of a 40 per

cent solution by volume, or 36·5 per cent by weight. A certain amount of methanol—about 12 to 14 per cent—is added, to prevent the polymerization of the aldehyde. Paraformaldehyde or paraform, a white crystalline powder, is a polymerized product, produced by evaporating solutions of formaldehyde. When heated, this substance evolves formaldehyde vapour and is used for disinfecting purposes.

Methanol is largely used as a solvent for varnishes, lacquers, and celluloid products. The dye industry requires considerable quantities both for solvent purposes, and for conversion into such compounds as dimethyl sulphate, and methyl iodide, both of which are used for introducing the methyl group into the complex organic dye molecules. The methyl radicle is of great value in the manufacture of organic substances for medicinal and photographic purposes, and is a constituent of many synthetic perfumes and flavours. Much of the crude methanol finds a use as fuel for stoves and lamps.

Formaldehyde

The uses of formaldehyde are many, and this compound is being increasingly employed in the chemical industries.

When mixed with phenols and similar compounds, and subjected to the action of alkali, heat and pressure, various synthetic resins are formed, known in commerce as bakelite, redmanol, condensite, etc. This field of industry takes a large quantity of formaldehyde.

Dilute solutions of formaldehyde have the property of hardening nitrogenous material such as gelatin, casein, etc., and on this account formaldehyde is of great value in the tanning and photographic industries.

It is also largely used as a disinfectant, deodorizer, and insecticide. Other minor uses are in the manufacture of medicinal compounds, in dyeing and textile printing, and in embalming and preserving.

THE CHEMICAL TREATMENT OF NATURAL GAS

Some of the chief developments in industrial chemistry during the last ten years have been based on gas reactions, in which gases have been utilized as the raw materials. The outstanding example is the synthesis of ammonia from hydrogen and nitrogen, by the Haber process, the success of which was one of the chief factors that enabled Germany to declare war in 1914. On the other hand, another process which perhaps helped to defeat Germany was the development in Canada of acetone and acetic acid manufacture from acetylene. Other processes using gases as raw materials are the manufacture of ethyl alcohol from ethylene present in coke oven gases, of isopropyl alcohol from propylene derived from the unsaturated gases evolved in the cracking of oils, also the production of gasoline from many natural gases, together with helium from the same source. Controversy has arisen in the United States over the rightful use of natural gas, whether it should be used as raw material for chemical processes or as domestic and industrial fuel. Whichever may be settled, there are many millions of cubic feet of natural gas which are being wasted in Canada, a commodity which would provide ample raw material for wealth producing chemical industries.

THE OXIDATION OF NATURAL GAS

When natural gas is burned in an unlimited supply of air or oxygen, carbon dioxide and water are the only products. The theory of the mechanism of combustion postulates the process as occurring in stages with the intermediate formation of complex compounds composed of water bound to the carbon atom of the methane molecule. Such compounds might be CH_3OH, $CH_2(OH)_2$ $CH(OH)_3$ and $C(OH)_4$.

It would be expected that these, on further oxidation and decomposition, would form HCOOH, HCHO, CO_2, CO and H_2.

If suitable conditions could be found, therefore, it should be possible to stop the oxidation halfway, and to isolate the more stable intermediate compounds, such as methanol and formaldehyde. Thus formaldehyde can be often smelled on lighting a natural gas burner in a cold furnace.

These compounds have been found qualitatively in the partial combustion of natural gas, and it is recognized that in the near future commercial processes will be established for their production in this way.

PATENTS COVERING THE OXIDATION OF NATURAL GAS

Several patents of processes for the production of formaldehyde and methanol have been taken out at various times. One of the earliest is that of H. S. Blackmore, U.S. Pat. 774824, 1904. According to this patent, methane is passed over substances such as ferroso ferric oxide, cupric or manganese oxides at 260° F. to produce methanol, or at 315° F. to give formaldehyde. He claims to get 125 pounds of methanol, or 118 pounds of formaldehyde, from 1,500 cubic feet of methane, which works out to be an efficiency of over 90 per cent. Considerable quantities of the oxides are used up, so that the process cannot be considered a catalytic one.

A somewhat similar German patent assigned to Glock, D.P.R. 107014, specifies the passage of equal volumes of methane and air over granulated copper on pumice heated to 800° F., with the resulting production of formaldehyde.

Fernekes in U.S. Pat. 1038546, 1911, employs a special form of combustion furnace, in which the oxidation is regulated so as to isolate the products of partial combustion.

Unruh U.S. Pat. 891,753, 1907, subjects mixtures of methane and air to the catalytic action of specific vegetable products, such as oak bark, at a temperature of 30° to 60° C., and claims high yields of formaldehyde, formic acid, and methanol. Another early patent was that of Lance and Elworthy, Brit. Pat. 7297, 1906, who claimed to oxidize methane by hydrogen peroxide with the addition of ferrous sulphate or monopersulphuric acid.

A number of inventors have attacked the problem by first chlorinating methane, separating the methyl chloride—which is one of the products of the reaction—and then hydrolysing the chloride to form alcohol. The chief patents on chlorination are those of Lacy, U.S. 1,111,842 and Can. Pat. No. 173,969, 1916, C. W. Bedford U.S. Pat. 1,245,553. J. Walter, German Pat. 222,919, 1909. A further list is given in U.S. Technical Paper 255, "The chlorination of natural gas," U.S. Bureau of Mines, 1921.

The second stage, the conversion of methyl chloride to alcohol, is also the subject of patents, Lacy, Can. Pat. No. 191,542, 1919, but the efficiency of the reaction is doubtful.

CANADIAN NATURAL GAS FIELDS

The chief gas fields in Canada are situated in southwestern Ontario and Alberta, and have been fully described, at least from a geological point of view, in several reports[1] issued by the Department of Mines. Two other reports[2] give more information from the chemical side.

In Ontario, the fields may be grouped into seven districts which yield the following approximate annual production:—

	Cubic feet per year
Oil Springs, Petrolia field	170 million
Tilbury and other Kent fields	8,000 "
Selkirk, Rainham, Dunville field	2,100 "
Brant-Onondaga field	60 "
Blackheath-Seneca field	500 "
Welland field	3,000 "
Toronto field	negligible

Chemical analysis of many gases from these fields, made by Messrs. Ellis, Bain, and Ardagh[2], show great similarity of composition, the average being about 80 per cent methane, 12 per cent ethane, and 8 per cent nitrogen. The following are typical analyses:—

County and well	Methane	Ethane	Nitrogen	Carbon dioxide	Hydrogen sulphide	Flow, cubic feet per day
Askew, Kent No. 2	84·4	10·8	4·5	.....	0·3	2,000,000
Lambton, No. 3	80·2	11·7	8·1	.....	.....	.....
Haldimand, No. 2	79·7	11·4	8·9	.....	.....	470,000
Welland, No. 2	80·0	12·9	7·1	.....	.....	1,000,000
York Scarboro tp	85·1	.....	13·2	1·6	.....	.....

More recent analyses of Ontario natural gases, carried out in this Department, follow:—

	Methane	Ethane	Nitrogen	Carbon dioxide
Dominion Natural Gas Co., well at Simcoe, Ont	80·3	7·6	11·8	0·3
St. Bernardin, Prescott co	66·2	.....	33·0	0·8

1. Petroleum and Natural Gas Resources of Canada, Vols. I and II, F. G. Clapp and others, Mines Branch Report No. 291, 1914.
The Oil and Gas Fields of Ontario and Quebec, Wyatt Malcolm, Memoir 81, Geological Survey, 1915.
Investigations in the Gas and Oil Fields of Alberta, Saskatchewan, and Manitoba, D. B. Dowling, S. E. Slipper and F. M. McLean, Memoir 116, Geolog. Survey, 1919.

2. Ontario Bureau of Mines 23rd Annual Report, 1914, pp. 237-273. Report on some Sources of Helium in the British Empire, by J. C. McLennan and Associates, Bull, No. 31, Mines Branch, 1920.

In Alberta, the fields may be considered under the following districts:—

	Approximate supply per year, cubic feet.
1. Medicine Hat field	7,300 million
2. Bow island field	13,000 "
3. Sweet Grass country, Milk River field	1,000 "
4. Suffield Brooks Bassano, Calgary field	1,200 "
5. Okotoks field	700 "
6. Wetaskiwin, Viking, Vegreville field	1,300 "
7. Athabaska field	300 "

The figures for the available supply are only very approximate, calculated from the estimated daily flow. They are larger too than the reported production, which was 8,230 million cubic feet, valued at $1,365,127, in 1919.

On an average, the Alberta gas is higher in methane than the Ontario gas, as shown by the following analyses:[1]—

Locality of well	Methane	Ethane	Nitrogen	Oxygen	Carbon dioxide
Medicine Hat city gas	90	3	5	0·1	0·8
Calgary and Bow island	91·3	.....	8·4	0·2	0·1
Pipe line, Okotoks gas, Dingman well—					
(i) Casing head gas	66·1	34·1	.....	0·49	0·25
(ii) Bottom gas	99·0	.....	.....	0·2	0·5
Viking	94·0	.....	5·0	0·4	0·6
Barnwell well No. 25	87	5·0	7·6	.....	.....
No. 9 and 10 and Southern Alberta	88·2	0·9	11·0	.....	.....

1. From Bulletin 31, Mines Branch, pp. 21 and 55.

The quantity of gas produced in New Brunswick, chiefly from the Stony Creek field, is small, and amounted to 6·82 million cubic feet in 1919.

A recent analysis of gas from this field, collected near Moncton, gave the following result:—

	Per cent.
Methane CH_4	80·0
Ethane C_2H_6	7·2
Carbon dioxide	none
Oxygen O_2	none
Nitrogen and rare gases N_2	12·8
Density	0·82

Very little natural gas has been found in Quebec, and no analyses are to hand.

Similarly, in British Columbia natural gas has been struck only at Port Haney, Pitt Meadows, on Pender island in the Gulf of Georgia, and recently near Kamloops.

The following analyses show that in two cases the gas is almost entirely nitrogen:—

	Methane	Ethane	Nitrogen	Carbon dioxide	Oxygen
Port Haney	22·5	0·15 (olefines)	68·3	1·2	7·5
Pitt Meadows	none	.....	99·0	0·5	0·5
Pender island	none	.....	99·0*		.....
Kamloops	96·8	absent	2·9	0·3	.....

*A recent analysis does not agree with this result.

The latter gas was analysed by the fractional distillation method, an apparatus for this purpose having been assembled during the year. Other Canadian gases should be analysed in this way, as this method gives the proportions of gasoline hydrocarbons present, and is far more valuable than the ordinary combustion analysis.

This brief survey shows that although there is comparatively little reliable information on the composition of Canadian natural gases, there are undoubtedly many which would serve as suitable raw material for chemical processes.

EXPERIMENTAL WORK IN THE MINES BRANCH LABORATORIES

After a survey of the patents and the scientific literature, the following lines of attack were decided upon:—

(I) Oxidation in the gaseous phase by passing methane and oxygen or air over various catalysts.
(II) Oxidation by ozone.
(III) Iodation, and subsequent hydrolysis.
(IV) The reaction between carbon dioxide and methane.
(V) The formation of ethylene from methane, and its subsequent oxidation.
(VI) Oxidation of methane in solution, and by liquid oxidizing agents.

The following pages outline briefly the results of the experiments on each of those methods.

Method I: Oxidation in the gaseous phase

A natural gas containing 80·3 per cent methane, obtained from a well at Simcoe, Ontario, by the courtesy of the Dominion Natural Gas Co., was used for the experiments. This gas, mixed with oxygen, after being purified, was led at varying flow rates over catalysts in a special furnace. The products of the reaction were absorbed in water or collected in a gasometer, and examined qualitatively and quantitatively for methanol, formaldehyde, formic acid, carbon dioxide, carbon monoxide, oxygen, and methane, by the usual methods.

The chief conditions which control the reaction are:—

(1) Ratio of methane to oxygen.
(2) Nature and form of catalyst.
(3) Temperature of catalyst.
(4) Time of contact, and space time yield.
(5) Effect of water vapour and impurities in the gases.
(6) Pressure conditions.

Several series of experiments were carried out, using gas mixtures containing two, three, five, and nine parts of methane to one of oxygen, respectively, at flow rates from three to fifteen litres per hour.

The catalysts employed were natural magnetite, ferric oxide, copper oxide, copper, silver oxide, silver, platinum, thorium oxide, and borosilicate glass, distributed on materials such as pumice, asbestos, and activated charcoal.

Tests were made with the catalysts maintained at temperatures between 150° and 500° C.

In some experiments the gases were thoroughly dried, in others water vapour was present.

Analyses showed that carbon dioxide was always a product of the reaction at temperatures over 250° C. Formaldehyde was detected at 500° C. with each catalyst, and in the case of the copper oxide catalyst at 400° C. It was never in sufficient amounts for a satisfactory quantitative measurement.

Method II: Oxidation by ozone

Several observers have studied the effect of ozone on hydrocarbons, especially Otto (Annales de Chimie et de Physique Series), Vol. 13, 1898, p. 109) and Drugman (Jour. Chem. Soc. 89, 1614, 1906).

They obtained traces of methanol, formaldehyde, and formic acid, when methane and ozonised oxygen were mixed, both at room temperature and at 100° C., but the amounts formed were very small, even though 200 litres of methane was used in some experiments.

Hauser and Herzfeld (Ber. 45, 3575, 1912) state that small amounts of methane are quantitatively oxidized to formaldehyde at ordinary temperatures.

Experimental.—A special ozoniser was constructed, and many experiments carried out to obtain a highly ozonised oxygen. The usual concentration of ozone was about 3 per cent by volume of the oxygen used.

This ozonised oxygen was mixed with natural gas, and the mixture led through a reaction tube at various temperatures, and finally through condensers and wash bottles.

In some experiments, activated charcoal, or platinized asbestos, was used in the reaction tube. No formaldehyde or methanol was detected.

The arrangement was then changed, and natural gas and oxygen passed through the ozoniser, in the proportion of 1:2 and at varying speeds. An oily liquid was formed in the ozoniser tube, which was found to be a mixture of polymerised aldehydes, methanol, acetone, and formic acid. The wash waters gave strong positive reactions for aldehyde and alcohol. Owing to experimental difficulties, and occurrence of explosions, only preliminary experiments were made. It seems one of the most promising methods of the oxidation if it can be controlled and prevented from going too far.

Method III: Iodation

Many investigations have been conducted on the chlorination of natural gas, and many patent processes are recorded. The U. S. Bureau of Mines Technical Paper 255, 1921, contains a good bibliography on this subject. Although the production of methanol by the hydrolysis of methyl chloride, one of the products of chlorination, is said to be difficult to carry out, one corporation in the United States has developed a process for the manufacture of methanol in this way, though at the present low prices it is not economically practicable.

The reaction between methane and iodine should be much more under control so that the mono-halide could be made the chief product. It would be expected, too, that methyl iodide would be more easily hydrolysed than the corresponding chloride.

Experimental.—Apparatus was arranged so that mixtures of natural gas and iodine vapour were passed over activated charcoal, with and without metallic catalysts. The products of the reaction were separated by condensation and water absorption.

The rates of flow, temperature, and nature of catalysts were varied, and in every experiment methyl iodide was obtained. The yields were small, however, never amounting to more than 5 per cent of the theoretical yield, assuming only the formation of the mono-halide.

The efficiency of the hydrolysis of methyl iodide with water, and with alkali solutions, was also studied, and was found to be far from complete at ordinary pressures. The best results with solutions of sodium hydroxide containing 40 gms. per litre showed only 38 per cent of the theoretical yield of alcohol.

Even though the difficulty of recovering and re-using the iodine could be successfully overcome the process could not be made economic.

Method IV: Reaction between methane and carbon dioxide

British Pat. No. 156,148, 1921, by H. Plauson and J. A. Vielle, specified the production of formaldehyde and methanol by passing a mixture of methane and carbon dioxide through a constricted metal tube of copper, silver, nickel, or alloys of these metals with aluminium, tin, or zinc, heated to several hundred degrees temperature.

Although carbon dioxide is only dissociated into carbon monoxide and oxygen at very high temperatures (2,500-3,000° C.) it is possible that in contact with heated metals some decomposition may take place at lower temperatures, and the products may react with the methane present giving the desired intermediate oxidation products.

Experimental.—Natural gas containing 80 per cent methane was mixed with carbon dioxide in various proportions, and the mixtures passed over copper gauze, platinized asbestos, or silver on asbestos, packed into a quartz tube. These catalysts were heated to various temperatures between 500° and 800° C. Gas analyses were made of the initial and final gases and the wash waters were examined for the presence of formaldehyde and methanol. No indication of these substances was obtained below 700° C. At 750-800° C. traces of formaldehyde were detected, and in three experiments qualitative measurements were carried out. The results showed that only about one-thousandth of the theoretical yield was formed.

Method V: Formation of ethylene from methane

It has been recently shown that ethylene may be readily oxidized to formaldehyde.[1]

Ethylene does not occur naturally, though it is present to the extent of 3 to 5 per cent in coal gas. The recent development of the manufacture of ethyl alcohol from ethylene derived from this source has received much attention. Other uses would undoubtedly be found if a cheap supply of ethylene was developed.

It is stated (De Boistesselin and Dubosc in "Rubber, its Production, Chemistry and Synthesis," Griffin, London, 1918) that ethylene can be formed by passing methane over carbon deposited on copper, heated to 400-450° C., though from the work of V. B. Lewis (Proc. Roy. Soc. 55, 99-107, 1894) on the decomposition of ethylene such a result does not seem probable. Some experiments were therefore undertaken to see if ethylene could be obtained in this way.

Natural gas was passed over copper gauze on which carbon black had been deposited at temperatures of 500° to 850° C.

Analyses of the initial and final gases were carried out. At 620° C. and at 845° C., 1·6 per cent and 2·8 per cent ethylene was found in the exit gas. Several experiments were made to determine the optimum flow rate and the space time yields. The best yield of ethylene obtained was 3·6 per cent of the volume of methane in the initial mixture. A much higher yield would be necessary for a commercial process to be successful, working along this line.

Method VI: Oxidation of methane in solutions

A special apparatus has been constructed to measure the solubility of methane in various solvents, and to carry out experiments on oxidation of this dissolved gas.

Only preliminary experiments have, as yet, been carried out.

The work on these various methods has been summarized very briefly, and results given in a few lines have been only obtained by a series of experiments and analytical work occupying many weeks and months. Although the data collected does not give promise of a successful industrial process, only the fringe of the investigation has really been covered. There remains to be studied further extensions of the methods already tried: the effect of catalysts such as uranium, molybdenum, and vanadium, which are being employed so much at the present time, and other methods based on new principles. Much more data must be collected on the equilibria between the lower hydrocarbons and their oxidation and decomposition products, without which no real progress can be made. There are many other problems connected with natural gas and with the hydrocarbon gases which will amply repay solution, such as the more efficient separation of gasoline, and production of carbon black, and the synthesis and utilization of the higher hydrocarbons, with the ulterior production of liquid paraffins, the higher alcohols, aldehydes, and acids, and all the valuable industrial products of which they are the raw materials.

[1] R. Willstatter and Bommer Annalen 422, p. 36046, 1920. Chem. Abs. 15, 1925, 1921.

II

CHEMICAL AND PHYSICAL CHARACTERS OF BENTONITE

E. A. Thompson and A. Sadler

An investigation of the chemical and physical characters of bentonite has been undertaken as part of a general study of this interesting material referred to by Mr. H. S. Spence, at page 15 of Summary Report of Investigations by the Mines Branch during 1920, and by Mr. Joseph Keele, at page 159 of the Summary Report of the Mines Branch for 1918.

As noted by Mr. Keele and by Mr. Spence in the reports above referred to, the variety of clay known as bentonite occurs at several localities in western Canada.

It differs in several respects from ordinary clay, and possesses properties which it is believed will make it of considerable economic importance.

It is described by Mr. Keele as follows: "When freshly exposed it varies from a light yellow to light olive green, with waxy lustre..... The clay is exceedingly fine-grained and has a soapy feeling when wet. It swells and forms a jelly-like mass in an excess of water." (1918 Summary Report, p. 159).

Its chief use, hitherto, has been as a filler in paper manufacture, but it has been suggested that it might be employed for any of the following purposes: (1) soap making, in which process it can actually replace a portion of the soap substance; (2) as a filler in rubber, textiles, leather, phonograph records, cordage, pressed and moulded insulations; (3) as an ingredient in gypsum and lime plasters; (4) in ceramics; (5) for replacing, in part, the bonding clay in abrasive wheels, graphite crucibles, chemical and electrical porcelain; (6) as an adhesive paste; (7) for dehydrating crude petroleum; (8) as a water softener, and base for massage creams; (9) in printer's ink; (10) as a substitute for fuller's earth.

Analyses.—Eight samples from five widely separated localities have been analysed, as follows:—

1. British Columbia—near Merritt. Top of an 8-foot bed.
2. British Columbia—near Merritt. Middle of an 8-foot bed.
3. British Columbia—near Merritt. Bottom of an 8-foot bed.
4. British Columbia—near Princeton.
5. Alberta—Camrose.
6. Alberta—Rosedale.
7. Wyoming, U.S., from the earliest discovered deposit.
8. Wyoming, U.S. (?), wilkinite, a trade name for bentonite, precise locality where found is not stated.

Analyses by Mr. A. Sadler upon air dried material gave the following results:—

	British Columbia Top	British Columbia Middle	British Columbia Bottom	Princeton	Camrose
H_2O at 105°C	3·92	6·64	6·09	7·71	5·94
H_2O over 105° C	4·28	4·92	4·73	3·24	3·70
SiO_2	69·70	67·04	69·60	68·60	60·78
FeO	0·23	0·23	0·22	0·32	0·43
Fe_2O_3	2·88	2·79	1·79	2·00	3·67
Al_2O_3	14·00	13·46	11·84	12·10	17·08
CaO	1·56	1·78	2·28	1·84	2·84
MgO	1·81	1·93	1·75	1·84	2·16
TiO_2	0·23	0·25	0·20	0·14	0·22
P_2O_5	0·014	0·09	0·09	0·17	0·033
Na_2	1·09	0·53	0·48	0·50	1·54
K_2O	0·29	0·22	0·14	0·23	0·46
SO_3	none	none	none	0·61	0·005
S	0·016	0·008	none	none	0·04
CO_2	0·04	0·02	0·73	0·17	0·11
Cl	trace	trace	trace	trace	trace
C	0·09	0·10	0·07	0·08	0·13
N	.....	.....	.....	.....	.....
	100·150	100·008	100·01	99·55	100·138

	Rosedale	Wyoming (I)	Wyoming(?) (II) Wilkinite
H_2O at 105°C	8·10	5·80	11·91
H_2O over 105° C	4·28	3·67	3·24
SiO_2	54·88	66·90	53·90
FeO	0·22	0·12	0·18
Fe_2O_3	4·10	2·80	3·21
Al_2O_3	19·92	15·26	18·00
CaO	2·22	0·46	1·00
MgO	2·83	2·26	3·61
TiO_2	0·25	0·11	0·65
P_2O_5	0·09	0·04	0·10
Na_2O	1·75	2·12	2·72
K_2O	0·26	0·42	0·21
SO_3	0·16	0·08	0·19
S	none	0·018	none
CO_2	0·37	0·05	0·21
Cl	trace	trace	trace
C	0·07	0·03	0·53
N	.....	.....	none
	99·50	100·138	99·66

The foregoing analyses were undertaken with four objectives in view:—

(*a*) To ascertain if any definite chemical relationship existed between the several samples.

(*b*) To calculate, if possible, a definite (mineralogical) formula for bentonite.

(*c*) To discover its origin.

(*d*) To have an accurate and complete record of its composition.

Comments—

(*a*) The silica content in these clays varies widely from 53·90 per cent in the wilkinite to 69·70 per cent in the British Columbia (top) sample. The wilkinite (53·90 per cent) and the Rosedale samples (54·88 per cent) do agree fairly closely, which, however, may be only a coincidence. The alumina varies from 11·84 per cent in No. 3 to 19·92 per cent in No. 6. These variations of the alumina and silica do not indicate a very definite relationship between the materials of the different deposits. Apart from these the amounts of the other constituents agree quite closely.

It was noticed that a high silica content is attended by a low alumina. The converse is also true.

(*b*) The calculation of a formula does not appear to be possible, at present. This may be accomplished after a microscopic examination has been made.

(*c*) Analysis does not give any definite information as to the probable origin of these clays as it does in the case of kaolin. The probability is that these were derived from highly siliceous rocks, possibly volcanic, which have undergone subsequent alteration.

(*d*) These analyses, it is believed, constitute the most complete record of composition of Canadian bentonite at present available.

ANALYSES OF CERTAIN AMERICAN BENTONITE CLAYS

The following analyses of similar material are published in Bulletin No. 2289 U.S. Bureau of Mines Reports of Investigations, 1921, entitled "Bentonite Supplies, Properties, and Uses," by Raymond B. Ladoo, Mineral technologist, U.S. Bureau of Mines.

It was deemed advisable to include these analyses, in order to afford comparison with the Canadian bentonite clays.

	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6	No. 7	No. 8
SiO	60·54	57·98	58·68	59·84	60·18	63·20	54·00	55·22
Al_2O_3	23·26	22·46	25·91	11·84	26·58	12·90	24·48	21·00
Fe_2O_3	3·92	3·80	3·97	3·26	.....	2·46	3·00	3·61
TiO_2	0·12	.....	0	.....	.....	0·11	.....	.....
CaO	0·59	1·92	1·45	2·90	0·23	0·82	2·08	4·94
MgO	2·19	3·24	1·49	2·32	1·01	2·09	2·75	3·04
K_2O	0·37	1·35	0	2·34	1·23	0·26	.....	.....
Na_2O	4·33		1·39	2·13		0·66	1·74	1·56
SO_3	.....	0·75	0·11	.....	.....	.....	.....	0·43
CO_2	.....	.....	tr.	.....	.....	3·50*	.....	tr.
Cl	.....	.....	0·10	.....	.....	0·20**	.....	0
P_2O_5	.....	.....	0·06	.....	.....	.....	0·71	.....
Ign. loss	.....	7·93	.....	10·50	.....	.....	.....	.....
H_2O	2·83	.....	6·84	.....	10·26	13·80	9·12	10·28
Total	98·25	99·43	100·00	95·13	99·49	100·00	97·88	100·08

*Sand, etc. **Water soluble.

No. 1. Yellow colloidal; Belle Fourche, South Dakota. Analyst, W. A. Selvig, U.S. Bureau of Mines.

No. 2. Yellow, colloidal; Medicine Bow, Wyoming. Analyst, A. G. VanEman, Owyhee Chemical Products Co., Chicago, Ill.

No. 3. White colloidal; near Barstow, California. Analysis from California Master Products Co., Los Angeles, California.

No. 4. White, fine-grained; Otay, San Diego county, California. Analysis from Otaylite Products Co., Los Angeles, California.

No. 5. Type material from near Rock Creek, Laramie Basin, Wyoming. Analyst, T. T. Read, Cassa Mining Co.

No. 6. From Big Horn Basin, Wyoming. Analyst, R. C. Wells, U.S. Geological Survey.

No. 7. Supposed bentonite, near Shelbyville, Tenn. Analyst unknown.

No. 8. Ardmorite, probably variety of bentonite, Ardmore. South Dakota. Analysis from the Refinite Co., Omaha, Nebraska.

Physical Properties

(*a*) *Absorptive Powers, for Water, Oils and Glycerine.*

One gram from each of the undermentioned samples was found to absorb the indicated number of grams of water. Clay and water in contact for 24 hours:—

British Columbia	1·53
Camrose	4·15
Rosedale	4·71
Wyoming	4·93
Wilkinite (?)	4·95

Experiments with oils and glycerine have not yielded any satisfactory results.

(*b*) *Specific gravity.*—Crude material, ground to 100 mesh, dried at 105° C. for 2 hours. Determination by pycnometer, gave the following results:—

British Columbia (composite)	2·44
Rosedale	2·72
Camrose	2·73
Wyoming	2·77
Wilkinite (?)	2·78

(*c*) *Fusion point.*—

	Cone No.	Temp.
British Columbia	15	1430° C.
Rosedale	14	1410° C.
Camrose	1	1150° C.
Wyoming	11	1350° C.
Wilkinite	10	1330° C.

(*d*) *Effect of heating on the swelling, and colloidal properties of the crude material.*—Determinations were made of the water remaining in the clay after it has been heated to the indicated temperatures for a period of one hour. It was also found that after heating to a certain temperature, which differed in the different samples, that the swelling property was destroyed, and that the other colloidal properties were diminished as the temperature increased. The results obtained are indicated below.

Percentage amount of water remaining in the clay.

	Before heating	450° C.	500° C.	550° C.	600° C.	700° C.
B.C. Composite	4·64	*4·04	3·94	.....	2·53	1·50
Camrose	3·70	.....	*3·60	.....	2·09	1·14
Rosedale	4·28	.....	.....	*4·17	.....	
Wyoming	3·67	.....	3·49	.....	*3·43	0·77

*Swelling property completely destroyed.

The foregoing results show that there is no fixed relation between swelling properties and the quantity of combined water present. In the B.C. sample, swelling was destroyed at 450° C., and in the Wyoming sample at 600° C., and yet they both retained the greater quantity of their combined water.

It was also found that after heating from 400° C. to 600° C., there remained only a very small quantity in suspension after standing in water for 24 hours. Heating to 700° C. destroyed the colloidal property altogether. Even at 700° C. combined water still remained in the clay.

ORE DRESSING AND METALLURGICAL DIVISION

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

The Ore Dressing and Metallurgical Division conducted, during the calendar year 1921, a considerable number of investigations involving extensive experimental and research work on Canadian ores and metallurgical products. Some of these investigations were completed and reported upon, and on others the work is being carried into 1922.

An idea of the extent and scope of the work of the testing and research laboratories can be obtained from a review of the list of ores and metallurgical products received for experimental purposes, which is given in tabular form following, under their respective test numbers. These may also be classified under the following investigations:—

1. THE LOW-GRADE COPPER ORES OF HIDDEN CREEK, ANYOX, B.C.

This investigation is described under test No. 140 of the Ore Dressing and Metallurgical Laboratories, and consisted of a series of flotation tests on a small scale, using the laboratory type apparatus, and also of larger tests, using the commercial size machinery. Various reagents were used, such as the Rex flotation reagents, oils, with and without other reagents, and the Alphabetical reagents. The larger scale tests were conducted on both the mechanical agitation and pneumatic types of flotation machines. A few of the larger scale tests remain to be conducted, but the results of the work to date show that the ore is amenable to concentration by flotation. A high recovery of the copper values was obtained in a good grade of concentrate.

2. THE FLIN FLON COMPLEX SULPHIDE ORE

This investigation is described fully under test No. 142 of the Ore Dressing and Metallurgical Laboratories, and consisted of a large number of flotation tests, using the laboratory type apparatus on the differential flotation of the minerals in this ore. The Flin Flon ore-body consists of three distinct mineralized zones—the hanging wall zone of disseminated ore, the centre zone of practically solid sulphides of iron, copper and zinc, and a footwall zone. This investigation covers the experimental and research work on the centre sulphide ore, and the object was to determine if the minerals could be separated into products suitable for reduction, or metallurgical treatment, into the metallic state. The shipments received consisted of drill cores crushed to about one-half inch size, from the exploration work on this property.

A microscopic examination of some of the larger pieces showed an intimate mixture of the three sulphides, with very little gangue mineral. Grinding to 200 mesh freed the greater part of the mineral constituents, and it was at this mesh that the most of the test work was conducted. The results of the test work showed that by differential flotation a copper concentrate, a zinc concentrate, and an iron tailing can be produced, but whether this can be accomplished on a large scale in practice has yet to be proven. It was also shown that the precious metal values in the ore are evenly distributed in all the sulphides, and that only 40 per cent of these values report in the copper concentrate. Besides the microscopic examinations made by the Mines Branch, examinations were made by Professor Wandke of Cambridge University, on the ore and concentration products, and his findings are given in the description of the test work under test No. 142.

3. THE CONCENTRATION OF GRAPHITE FROM THE NORTH AMERICAN MINE

This investigation is described fully under test No. 143 of the Ore Dressing and Metallurgical Laboratories, and consisted of a number of experimental tests, run under different conditions and various methods of procedure, to determine which would be the most desirable and practical for the recovery of the greatest proportion of large flake of a commercial grade, consistent with a high recovery of the graphite in the ore. A method of concentration was proven, in which these points were determined, adaptable to this particular ore.

4. THE CONCENTRATION AND SEPARATION OF THE MINERALS IN THE LEAD-ZINC-SILVER ORES OF KIMBERLEY, B.C.

This investigation is described under tests Nos. 144 and 145 of the Ore Dressing and Metallurgical Laboratories, and consists of the experimental work on two shipments of ore—one that is known as carbonate ore, and the other as sulphide ore. These ores are typical of the district, the carbonate ore being found in the upper portions of the ore-body, and the sulphide ore at greater depth. The carbonate ore consisted chiefly of lead carbonate, coated and locked up in a mass of iron oxides, formed by the oxidation of the iron sulphides. The sulphide ore was an intimate mixture of lead, zinc, and iron sulphides, the gangue minerals representing less than 10 per cent of the ore. On the carbonate ore experimental test work was conducted by sulphidizing previous to flotation and table concentration of the flotation tailing, and also by table concentration followed by sulphidizing and flotation of the table tailing. Volatilization tests were also conducted to determine whether the ore was amenable to this process.

By these methods a recovery of between 70 per cent and 80 per cent of the lead values, and 40 per cent of the silver values, is obtained.

On the sulphide ore, a series of tests were conducted, using various flotation reagents, to obtain a differential flotation of the lead, zinc, and iron sulphides. A lead concentrate was first obtained, and by changing conditions of the pulp, a zinc concentrate was next obtained, leaving a tailing of iron sulphides and gangue mineral. The lead and zinc products were of commercial grade, with good recoveries of the metals in these products.

5. SOME COMPARATIVE TESTS ON THE COBALT SILVER SAND TAILINGS

In continuation of some extensive experiments made in 1917 on the use of Canadian wood oils for the concentration of the values in the Cobalt slime tailing, a few experiments were conducted on the use of the Rex, Alphabetical and other reagents, on a shipment of sand tailing from one of the Cobalt mines. The results of this work are described under test No. 146 of the Ore Dressing and Metallurgical Laboratories.

6. THE MILLING OF ASBESTOS ROCK

This investigation is described under tests Nos. 147, 149, and 152 of the Ore Dressing and Metallurgical Laboratories, and consisted of a number of milling tests on the "slip fibre" variety of asbestos rock from the Boston mine, East Broughton, Que., and on the "cross fibre" variety from Black Lake, Que. The investigations carried on during the calendar year were confined to the use of the conical ball mill for the crushing of the rock and liberation of the fibre. The use of Cyclone and Jumbo fiberizers in the present practice in the asbestos districts is found to be costly as to power consumption, maintenance, and repair. The violent action in these machines as operated for the crushing of the rock and liberation of the fibre is found to be more or less destructive of the long fibre. It was found that the conical ball mill, operated under favourable conditions and equipped with suction take-off for the removal of the fibre when liberated, could be used in the milling of the rock, giving very satisfactory results. In conducting these milling tests, other points of interest were brought out, which are noted in the description of the investigations.

As one of the chief items of cost in the milling of asbestos rock is that of drying the ore for the present practice, and as this cost is increased greatly during the winter and spring seasons, it is the intention to continue the investigations on the wet grinding of the rock. If the dry milling practice could be dispensed with, and a wet process evolved to treat the rock successfully, it would eliminate the dust, resulting in much more pleasant conditions for the workmen and for the inhabitants of the districts in the vicinity of the mills.

7. THE CONCENTRATION OF THE LOWER GRADE COPPER-NICKEL ORES

This investigation is described under tests Nos. 148, 155, and 160 of the Ore Dressing and Metallurgical Laboratories, and consists of a number of table concentration and flotation tests for the elimination of the gangue and the concentration of the copper-nickel minerals in a higher grade product for further metallurgical treatment. In the case of the experimental work conducted under test No. 148, it was desired that the gangue minerals be eliminated and the metallic minerals of copper, nickel, and iron concentrated into a product suitable for the refining process of the Burrows Refining Co., who are conducting investigations on the manufacture of certain alloy steels. In the case of the experimental work conducted under test Nos. 155 and 160, the test work was confined to the flotation of the lower grade ores to produce a high-grade copper-nickel product with a high recovery of these values. It was found that the ores are amenable to concentration by flotation, and a copper-nickel product can be obtained

with a base metal value the equivalent of the blast furnace matte, and with a good recovery of the copper-nickel values in the ores. In connection with the investigation, a point which has not been proven is whether the precious metal values are contained in the concentrate or the tailing, and further work is being conducted to determine where these values report.

8. THE CONCENTRATION OF GARNET FROM ITS ROCK

This investigation is described in detail under tests Nos. 150 and 158 of the Ore Dressing and Metallurgical Laboratories, and consists of experimental work on samples from the vicinity of Bancroft, Ont., and from Depot Harbour, Ont., for the determination of garnet content, percentage of recoverable garnet, and methods of concentration and separation of the garnet from the gangue. In the case of the deposits at Depot Harbour, considerable attention was given to the work on the samples, which were taken systematically, and represented a large deposit situated favourably from a shipping standpoint for the disposal of the concentration products, and on account of the high percentage of recoverable garnet. It was found that the garnet could be concentrated and separated from the gangue, and recovered in marketable products, suitable for the trade, with high recoveries of the garnet content in the rock. Methods for its recovery were worked out, with a possible saving of certain mica products as a by-product from the operations.

9. THE RECOVERY OF BRASS FROM FOUNDRY REFUSE

The experimental work on this investigation is described under test No. 151 of the Ore Dressing and Metallurgical Laboratories, and consists of tests made for the concentration and separation of the brass from the slag, sweepings, and other refuse collected around brass foundries. It was found that it would be possible for many of the brass foundries to install a small unit, consisting of a ball mill, classifier, and table, to recover the brass from their refuse, and use the concentrated product in subsequent remelts.

10. THE METALLURGICAL TREATMENT OF THE GOLD ORES FROM THE RICE LAKE DISTRICT, MANITOBA

The experimental work on this investigation is described under tests Nos. 141 and 153 of the Ore Dressing and Metallurgical Laboratories. It consists of a series of tests to determine methods applicable to this particular type of gold ore, which contains copper minerals. It was found that the ore could be treated for the recovery of gold, with a high extraction of the precious metals, and also for the recovery of the copper, as a by-product of the milling operations. This report, with previous investigations on the ore from Little Rice lake, which is described under test No. 135 of the Summary Report for 1920, constitutes the experimental test work undertaken by the Department on the gold ores from the south-eastern section of Manitoba. The Report of the Ore Dressing and Metallurgical Laboratories, test No. 132, describes the experimental test work

on the gold ore from the Herb Lake district, in the northwestern section of the province. The ore from this district contains chalcopyrite, arsenopyrite, pyrite, sphalerite, and galena, and in this respect is somewhat different from the ordinary gold ores and requries a different method of treatment. It was found that the ore was amenable to treatment, described under test No. 132 in the Summary Report for 1920.

11. CONCENTRATION TESTS ON SOME MOLYBDENITE ORES

The concentration of the molybenite ores of Canada has been fairly well covered by the experimental work carried on in the Ore Dressing and Metallurgical Laboratories during the war years, when there was a brisk demand for the metal for the manufacture of tool-steel and for other munition purposes. Since that time the industry has been more or less dormant, the production in Canada falling off to practically nothing. Interest during the year has been revived somewhat, due to the research conducted to find new uses for industrial purposes. It has been determined that the use of fractional percentages of the metal, that is, quantities up to one per cent, is beneficial to most steels. With a revival of the steel industry it is logical to assume that there will be a revival of the molybdenite industry of Canada. The experimental work conducted during the year was on ores representing two of the most promising deposits in Ontario and Quebec, and it was found that these ores were amenable to concentration, with a high recovery of the molybdenite values in a high grade concentrate. The test work is described under tests Nos. 154 and 157 of the Ore Dressing and Metallurgical Laboratories.

12. THE RECOVERY OF THE VALUES IN A SILVER LEAD ORE FROM ARROWHEAD, B.C.

The experimental work on this ore is described under test No. 156 of the Ore Dressing and Metallurgical Laboratories. The ore is a lead carbonate carrying silver values. During the year some concentration tests were made by tabling and flotation. Further work is being conducted and it will be reported on during the new year.

13. THE CONCENTRATION OF THE RADIO-ACTIVE MINERAL EUXENITE

The occurrence of this mineral in the feldspar quarry of lot 13, concession V, of the township of South Sherbrooke, county of Lanark, in the province of Ontario, has been reported on by Miller and Knight, who visited the property in June, 1917. The report was published in the American Journal of Science, Volume XLIV, September, 1917, and contains a description of the occurrence as seen at that date. The mineral was identified as euxenite by the Imperial Institute, London, England, who made the analysis which is given in the report. In the operation of the quarry since that date more of the mineral has been exposed, and shipments have been made to the Department for experimental work on the concentration of the mineral from the feldspar. It was found that due to its specific gravity, being almost 5, and that of feldspar between 2·5 and 2·75, there was no particular difficulty in making a separation of the minerals. The experimental work is described under test No. 159 of the Ore Dressing and Metallurgical Laboratories.

14. THE USES OF FLOTATION REAGENTS MANUFACTURED IN CANADA

During 1917, the Division of Ore Dressing and Metallurgy, in co-operation with the Forest Products Laboratories of Canada, Forestry Branch, Montreal, started an investigation on the use of Canadian wood oils for ore flotation. This investigation was undertaken at the request of the Cobalt mining interests, and was chiefly confined to the cobalt-silver ores. At that particular time some difficulty was being experienced by the mining companies using the flotation process for the recovery of valuable minerals in their ores, in obtaining an adequate supply of imported pine oil. The investigation followed along two lines, namely, the production in Canada of a satisfactory grade of pine oil in sufficient quantities from Canadian resinous wood waste, or the possibility of finding a substitute for pine oil. The investigation showed that it was impossible to produce at a profit a satisfactory grade of pine oil, on account of the low yield from Canadian resinous wood waste, so that this line of investigation was eliminated, and attention paid to finding a substitute. The most likely substitute was the hardwood oils obtained from the destructive distillation of Canadian hardwoods, which could be produced as a by-product. It was found that as far as the Cobalt ores are concerned, after numerous experiments on a small scale in the Ore Dressing and Metallurgical Laboratories, and on a large scale of several days run in one of the Cobalt mills, that results equally good as those obtained by the use of pine oil could be obtained.

This investigation was discontinued on account of more important work in connection with the war minerals, but is now being continued on other ores where flotation is used to recover the valuable minerals. The investigation will not be confined to the hardwood oils, or the distillation products from hardwoods, but will cover the use of the mineral oils and other reagents manufactured in Canada, or which could be produced at a profit. A questionnaire has been submitted to the mining companies asking for their co-operation in stimulating the use of Canadian-made products, and for data with respect to the quantity of oils and reagents used, costs, etc. As soon as this information is collected, and samples of the various flotation feeds are obtained, the experimental work will be conducted, to determine if Canadian products cannot be substituted in many cases for those imported.

15. THE RECOVERY OF THE METALS OF THE PLATINUM GROUP FROM PLACER CONCENTRATES

An investigation was commenced in 1919, on the recovery of the platinum group metals from placer concentrates, but due to the limited supply of material submitted for test purposes, and the small quantity of these metals in the sample, little progress was made. The test work conducted was described in the Summary Report for 1919, under test No. 109 of the Ore Dressing and Metallurgical Laboratories. A larger sample has been received, and the investigation is being continued.

Some experimental work was conducted by making fusion tests on the placer concentrates, by fluxing them with limestone, silica, and charcoal, and adding sufficient galena to the charge to collect the precious metals in a lead button. The results of the work are given under the report of the work of the Chemical Laboratories of the division.

II

LIST OF ORES AND METALLURGICAL PRODUCTS ON WHICH EXPERIMENTAL TEST AND RESEARCH WORK WAS CONDUCTED

In the following table is given a list of the ores and metallurgical products received, with the test number, class of ore or metallurgical product, source of shipment, shipper, and weight of shipment, on which experimental test and research work was conducted.

Test No.	Ore, or metallurgical product	Source of shipment	Shipper and address	Weight of shipment
				lbs.
140	Copper.....	Hidden creek, No. 2 mine.	Granby Consolidated Mining, Smelting & Power Co., Ltd., Anyox, B.C.	50,560
141	Gold.....	Rice Lake, Manitoba.	Pan Extension Gold Mines, Ltd., Winnipeg, Man.	10
142	Copper.....	Flin Flon mine, Manitoba.	Mining Corporation of Canada, Ltd., Toronto, Ont.	250
143	Graphite.....	North American mine, Buckingham, Que.	H. P. H. Brumell, Buckingham, Que.	50
144	Lead-silver.....	North Star mine, Kimberley, B.C.	Thompson & McKinney, Kimberley, B.C.	370
145	Lead-zinc-silver	North Star mine, Kimberley, B.C.	Thompson & McKinney, Kimberley, B.C.	350
146	Silver.....	Coniagas mine, Cobalt, Ont.	Coniagas Mines, Ltd., Cobalt, Ont..	2,000
147	Asbestos.....	Boston mine, E. Broughton, Que.	Asbestos Mines, Limited, Montreal, Que.	500
148	Copper-nickel...	Strathcona mine, Sudbury, Ont.	Burrows Refining Co., Ltd., Ottawa, Ont.	48,000
149	Asbestos.....	Boston mine, E. Broughton, Que.	Asbestos Mines, Limited, Montreal, Que.	26,000
150	Garnet.....	Lot 7, con. XI, Faraday tp., Hastings Co., Ont.	Robt. A. Bryce, Toronto, Ont.....	100
150	Garnet.....	Lots 29, 30, 31 and 32, con. XIV, XV, Ashby tp., Lennox and Addington co., Ont.	Robt. A. Bryce, Toronto, Ont.....	100
151	Brass foundry refuse.	Ontario Specialties, Ltd., Ottawa, Ont.	Ontario Specialties, Ltd., Ottawa, Ont.	16,520
152	Asbestos.....	Black Lake, Que.....	Black Lake Asbestos & Chrome Co., Black lake, Que.	20,000
153	Gold.....	Rice Lake, Man.....	Pan Extension Gold Mines, Ltd., Winnipeg, Man.	2,000
154	Molybdenite....	Lots 3 and 4, con. XX, Cardiff tp., Haliburton co., Wilberforce, Ont.	W. E. Joiner, Toronto, Ont.....	800
155	Copper-nickel...	No. 3 mine (Frood)...	International Nickel Co., Copper Cliff, Ont.	100
156	Silver-lead.....	Arrowhead, B.C.....	Mansfield Mining Co., Mansfield, Washington, U.S.A.	220
157	Molybdenite....	Lots 53 and 54, range X, Masham tp., Ottawa co., Indian lake, Masham, Que.	John Bain, Ottawa, Ont.....	100
158	Garnet.....	Depot Harbour, Ont..	Robt. A. Bryce, Toronto, Ont.....	13,500
159	Euxenite.....	S. Sherbrooke tp., Maberley, Ont.	Orser-Kraft Feldspar, Ltd., Perth, Ont.	1,600
160	Copper-nickel...	Murray mine, Nickelton, Ont.	British America Nickel Corporation, Nickelton, Ont.	200

A description of the experimental test and research work conducted is given under the heading of the investigation, with the corresponding test numbers, as a sub-heading, so that the test numbers will not be in order as denoted above, in the tabular form. In some cases an investigation will cover three test numbers.

III

REPORTS ON THE INVESTIGATIONS CONDUCTED

The reports on the investigations conducted by officers of the Division cover a wide range of metallurgical problems. The investigations were assigned to the technical officers who, by experience, training, and knowledge of the subject, were most fitted to carry out the particular investigation. Those investigations entailing experimental test and research work along flotation lines were assigned to C. S. Parsons. Others, in which test and research work on milling and other ore dressing and metallurgical problems was required, were assigned to R. K. Carnochan. Investigations with respect to the recovery of the precious metals, including those of the platinum group, were assigned to H. C. Mabee. These technical officers were assisted by R. J. Traill and B. P. Coyne in carrying out certain phases of the work which they were specially capable of, due to their knowledge of physical and chemical reactions. All investigations were under the direction of the Chief of Division, who, in consultation with the technical officers of the division, outlined the procedure to be followed in conducting the test and research work.

(1)

THE LOW GRADE COPPER ORES OF HIDDEN CREEK, ANYOX, B.C.

R. K. Carnochan and C. S. Parsons

Test No. 140

A carload of copper ore, shipping weight 50,760 pounds, was received at the Ore Dressing and Metallurgical Laboratories, September 4, 1920, from the Granby Consolidated Mining, Smelting and Power Company, Anyox, B.C. The ore was a representative shipment of the lower grade from Hidden Creek No. 2 mine, and consisted of chalcopyrite and iron sulphides in a greenstone gangue.

The purpose of the experimental work on this ore was to determine its adaptability to concentration by flotation methods; to obtain some comparative data on the use of oils, the Rex, Alphabetical, and other flotation reagents, and also on the use of the pneumatic and mechanical agitation types of flotation machines.

A number of small laboratory tests had been made on similar ore by R. E. Peterson, using certain oils and the addition of lime, giving very satisfactory results. The results of two of these tests are given below, as representative when making low and high grade concentrate:—

Test No. 913:

Amount tested: 500 grams.
Apparatus used: Janney test machine.
Reagents used: ¾ pound per ton of 50 per cent mixture of coal tar creosote and hardwood creosotes and tars; 2 pounds of lime per ton.

Product	Weight grams	Analysis per cent				Per cent of metals in products			
		Cu.	Fe.	S.	Insol.	Cu.	Fe.	S.	Insol.
Heads	500	2·30	25·3	19·0	34·8	100·0	100·0	100·0	100·0
Concentrate	133	7·61	36·7	37·4	8·6	88·1	38·4	52·0	6·7
Middling	85	1·04	27·2	20·7	28·8	7·7	18·4	18·4	14·5
Tailing	282	0·17	19·5	10·0	47·4	4·2	43·2	29·6	78·8

This test indicates a copper recovery of 93·3 per cent on retreatment of the middlings, with a concentration ratio of 3·54 to 1.

Test No. 916:

Amount tested: 500 grams.
Apparatus used: Janney test machine.
Reagents used: ¾ pound per ton hardwood creosotes and tars; 8 pounds per ton of lime.

Product	Weight grams	Analysis per cent				Per cent of metals in products			
		Cu.	Fe.	S.	Insol.	Cu.	Fe.	S.	Insol.
Heads	500	2·30	25·3	19·0	34·8	100·0	100·0	100·0	100·0
Concentrate	69	12·66	27·0	23·8	17·4	78·3	14·8	17·4	7·1
Middling	53	2·93	24·7	18·3	27·2	13·9	10·4	10·3	8·5
Tailing	378	0·23	24·8	18·0	38·0	7·8	74·8	72·3	84·4

This test indicates a copper recovery of 91·5 per cent on re-treatment of the middling, with a concentration ratio of 6·02 to 1.

These tests show a very satisfactory separation of the copper minerals from the iron pyrites and the gangue, and that this ore can be successfully treated by the flotation process, by the use of oils, with the addition of lime, yielding a good recovery and an economical ratio of concentration.

Small Scale Laboratory Tests Using the Rex Reagents

A series of small scale tests were made on the laboratory type flotation machines, using the Rex flotation reagents. The results of these tests are given in the following table:—

Test No.	Heads per cent Cu.	Concentrates per cent Cu.	Middlings per cent Cu.	Tails per cent Cu.	Concentration ratio	Recovery
27	1·97	9·15	.....	0·27	1 : 5·5	88·6
28	1·97	17·30	.....	1·15	1 : 20·0	44·0
29	1·97	13·62	.....	0·58	1 : 8·8	74·0
30	1·97	11·67	.....	0·57	1 : 7·7	75·4
31	1·97	10·45	.....	0·35	1 : 6·1	85·4
32	1·97	15·80	.....	0·72	1 : 11·9	66·7
33	1·97	12·30	5·00	0·43	1 : 8·3	79·0
34	1·97	14·45	7·75	0·75	1 : 12·6	62·9
37	1·97	11·10 7·44	1·96	0·25	1 : 6·1	87·7
45	1·75	14·30	5·75	1·20	1 : 28·8	31·9
47	1·75	10·05	.....	0·52	1 : 7·9	73·8
49	1·75	17·20	.....	0·90	1 : 19·6	50·7
58	0·90 (tails No.49)	6·76	1·89	0·40	1 : 15·8	53·6 77·1 =total rec'y No. 49 & 58.
70	1·08	7·95	2·15	0·45	1 : 13·4	59·7
71	1·08	13·00	1·50	0·15	1 : 19·4	84·6
72	1·08	7·55	1·45	0·25	1 : 9·1	77·9
2*a*	2·25	7·20	.....	0·40	1 : 3·7	87·0
3*a*	2·25	7·60	.....	0·42	1 : 4·2	87·6
4*a*	2·25	10·63	3·10	0·70	1 : 6·4	73·5
5*a*	2 25	10·35	2·50	0·25	1 : 5·1	90·7
6*a*	2·25	8·55	1·15	0·15	1 : 4·0	95·0
8*a*	2·25	13·50	2·60	0·12	1 : 6·2	95·5
14*a*	2 46	18·15 15·06	1·28	0·31	1 : 6·2	88·4
15*a*	2·25	19·64	2·97	0·39	1 : 11·4	78·9

The above tests were run under the following conditions, except where differences are noted:—

Janney laboratory flotation machine used, 500 grams of Anyox ore ground to 100 mesh, mixed in pebble mill 5 minutes, agitated in machine 5 minutes, and floated for 8 minutes.

Differences in procedure in running tests :—

Test No.	——
30	No mixing.
31	200 mesh.
32	200 mesh.
33, 34	Floated 8 minutes for concentrates and 8 minutes more for middlings.
37	Ran two 500-gram charges, floating 15 minutes; re-ran combined concentrates, agitating 5 minutes and floating 8 minutes for first concentrates and 8 minutes more for second concentrates.
45	Ore, −60. Floated 8 minutes for concentrates and 5 minutes more for middlings.
47	Ruth machine. Three charges of 1,000 grams, −60 each. Floated 5 minutes—products mixed to form one concentrate and one tailing.
49	Ore, −60. Three charges of 500 grams run—products mixed to form one concentrate and one tailing.
58	500 grams tailings test No. 49, −100. Floated three minutes for concentrates and three minutes more for middlings.
70	Callow machine. 500 grams ore, −10, ground 2 hours in ball mill, floated for 13 minutes.
71	Callow machine. 500 grams ore, −10, ground 1 hour in ball mill, floated for 12 minutes.
72	Callow machine. 500 grams ore, −10, ground one-half hour in ball mill, floated for 12 minutes.
2*a*	No mixing, agitated 3 minutes, floated 20 minutes.
3*a*	" " 3 " 20 " 200 mesh.
4*a*	" " 3 " 15 " 200 "
5*a*	" " " 20 " 200 "
6*a*	Size of ore not stated, no mixing, agitated, floated for 20 minutes.
8*a*	Size of ore not stated, no mixing, agitated 3 minutes, floated for 15 minutes.
14*a*	No mixing, agitated 3 minutes, floated 5 minutes for 1st concentrate, then floated for rougher concentrate. Rougher concentrate re-cleaned giving 2nd concentrate and middling.
15*a*	Ore 250 mesh, mixed for 15 minutes, times of agitating and floating not stated, rougher concentrate made and this re-cleaned, giving concentrate and middling.

Small Scale Laboratory Tests.

Test	Heads			Concentrate			Middling			Tailing			Total Extn. 75% recvy. mids.	Reagents added to ball mill				Reagents added to machines							Machine used	Pulp acid or alkaline
	Charge grams	Anal. % Cu	Mesh	Wt. gms.	Anal. % Cu.	Recvy. %	Wt. gms.	Anal. % Cu.	% of Metal	Wt. gms.	Anal. % Cu.	Loss %		Lime	$NaCO_3$	NH_4OH	Neutral Alkaline	Lime	$NaCO_3$	NH_4OH	X.Y.	Rex.	Other Oils	Other reagents		
11	990	2·20	−100	278	7·15	91·1	121	0·91	5·17	591	0·14	3·80	.....	0	0	0	N	0	0	0	1·0	....	....	....	R	N
15	1,000	2·20	−100	197	9·26	81·4	105	2·31	10·8	698	0·25	7·80	.....	½	..	....	N	..	..	..	0·5	....	....	....	R	N
14	1,000	2·20	−100	120	13·5	73·4	160	2·8	20·3	718	0·14	6·30	87·6	1	..	....	N	..	..	..	0·5	....	....	....	R	N
12	992	2·22	−100	249	8·13	91·0	125	1·13	5·10	618	0·10	2·80	.....	2	..	....	N	..	..	..	1·0	....	....	....	R	N
16	1,000	2·25	−100	87	21·5	83·4	68	3·08	9·06	855	0·20	7·60	.....	4	..	....	A	..	..	..	0·5	....	....	....	R	A
17	1,000	2·20	−100	104	19·4	87·7	59	2·40	6·10	837	0·17	6·2	.....	4	..	....	A	..	..	..	1·0	....	....	....	R	A
19	1,000	2·20	−100	212	8·8	85·0	84	2·56	9·4	708	0·18	5·6	.....	..	4	....	A	..	..	..	1·0	....	....	....	R	A
20	1,000	2·20	−100	85	20·8	81·0	69	3·44	10·9	849	0·22	8·2	.....	1	..	....	N	3	..	..	1·0	....	....	....	R	A
21	1,000	2·67	−100	42	15·78	24·8	104	3·45	18·1	816	1·88	57·1	.....	4	..	....	A	..	..	..	....	1·0	....	....	R	A
22	1,000	3·0	−100	46	14·21	21·6	100	9·72	32·0	854	1·65	46·4	.....	1	..	....	N	..	..	..	....	1·0	....	....	R	N
24	1,000	2·6	−100	92	7·92	27·9	207	2·87	22·8	693	1·84	49·3	.....	2	..	....	N	..	..	..	....	1·0	....	....	R	N
25	1,000	2·4	−100	274	7·83	89·0	173	0·70	5·0	542	0·27	6·0	.....	..	4	....	A	..	..	..	....	1·0	....	....	R	A
26	1,000	2·6	−100	196	11·0	83·8	195	1·55	11·6	607	0·20	4·6	.....	..	2	1·0	A	..	..	..	....	1·0	....	....	R	A
27	1,000	2·7	−100	315	7·70	90·8	178	0·71	4·6	508	0·26	4·6	.....	..	2	0·5	N	..	2	..	....	0·5	....	....	R	A
28	1,000	2·7	−100	340	7·60	93·0	81	0·70	2·0	573	0·18	4·76	.....	..	2	0·5	N	..	2	..	0·5	....	5c.c. Pine oil.	....	R	A
13	1,000	2·2	−100	239	7·93	84·2	110	1·60	.....	650	0·30	.....	.....	..	..	....	N	..	..	..	....	....	1·0 Fuel oil.	....	R	N
29	1,000	2·2	−100	192	12·5	88·6	113	1·68	7·0	704	0·17	4·4	.....	4	..	....	A	..	..	..	....	....	·25 P.O. 1 ·50F.O.	....	R	A
30	1,000	2·7	.....	180	13·68	90·5	55	1·87	1·03	764	0·20	1·53	.....	4	..	....	A	..	..	..	1·0	....	....	....	R	A
31	1,000	2·7	.....	223	11·0	89·6	94	1·04	0·97	684	0·28	1·91	.....	..	2	1	A	..	..	..	....	1·0	....	....	R	A
32	1,000	2·65	.....	243	8·42	76·85	269	1·88	5·06	472	0·20	0·94	.....	..	2	1	A	..	..	..	....	1·0	....	....	R	A
34	1,000	2·62	.....	252	9·70	93·1	93	0·58	2·2	645	0·18	4·4	.....	..	..	....	A	4	..	..	1·0	....	....	....	C	A
35	1,000	2·76	.....	595	8·27	91·6	154	2·22	6·3	1,206	0·09	2·0	.....	..	..	....	A	4	..	..	1·0	....	....	....	C	A
36	1,000	.....	.....	208	5·2	.....	168	1·5	.....	597	0·30	.....	.....	4	..	....	A	..	..	..	....	....	1·0 ketone residue.	....	R	A
37	1,000	.....	.....	112	11·5	86·4	47	1·8	5·0	821	0·15	8·05	.....	4	..	....	A	..	..	..	·75	....	....	....	R	A
39	1,000	1·60	.....	152	4·49	42·8	153	1·38	13·2	683	1·02	44·0	.....	..	2	1	A	..	..	..	....	1·0	....	....	R	A
38	.....	.....	.....	299	4·6	85·6	.....	.....	.....	.....	.....	.....	.....	..	..	....	....	..	..	..	....	....	....	....	....	....

Test No. 38:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 60 mesh.
Reagents used: 2·5 c.c. Rex reagent dissolved in carbon bi-sulphide.
Results: The first concentrate came up in a very dirty looking and sticky froth. The addition of soda ash with the addition of more Rex reagent improved the results.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate No. 1	107	7·20	7·70	48·1
Concentrate No. 2	192	3·12	5·99	37·4
Tailing	701	0·33	2·31	14·5
Heads	1,000	1·60	16·00	100 0

Test No. 40:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 100 mesh.
Reagents used: 8 pounds lime per ton added to ball mill; a few c.c. of the acid soluble portion of resin pitch in H_2SO_4 was added to cell.
Results: A froth consisting of watery looking bubbles heavy with copper mineral formed.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	138	9·12	12·58	81·5
Middling	154	1·25	1·93	12·5
Tailing	708	0·13	0·92	6·0
Heads	1,000	1·54	15·43	100·0

Test No. 41:

Amount tested: 1,000 grams.
Apparatus used: Ruth machine.
Degree of crushing: 60 mesh.
Reagents used: The insoluble portion of resin pitch after dissolving in caustic soda was taken up with ammonia.
Results: None.

Test No. 42:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 60 mesh.
Reagents used: 1 pound per ton of NH_4OH and 4 pounds per ton of soda ash added to ball mill; resin pitch dissolved in H_2SO_4 and the acid soluble portion was distilled to remove any oil.
Results: Good, but it required a lot of this reagent to form a stable froth.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	164	6·22	10·20	66·4
Middling	145	2·85	4·13	26·9
Tailing	691	0·15	1·04	6·7
Heads	1,000	1·54	15·37	100·0

Test No. 43:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 60 mesh.
Reagents used: 1 pound per ton of NH_4OH and 4 pounds per ton of soda ash added to ball mill; 1 pound per ton of double strength solution of Rex reagent from which the acid soluble portion was extracted.
Result: The action was similar to Rex reagent.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	156	8·05	12·56	78·3
Middling	171	1·45	2·48	15·4
Tailing	673	0·15	1·01	6·3
Heads	1,000	1·60	16·05	100·0

Test No. 44:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 60 mesh.
Reagents used: 1 pound per ton NH_4OH and 4 pounds per ton soda ash added to ball mill; Rex reagent No. 12 was separated with ether and alcohol, the water soluble portion of this was used.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	138	8·00	11·04	72·0
Middling	165	1·63	2·69	17·6
Tailing	697	0·23	1·60	10·4
Heads	1,000	1·54	15·33	100·0

Test No. 45:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 60 mesh.
Reagents used: Same as test No. 44, but the ether soluble portion of reagent used.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	88	12·77	11·24	74·8
Middling	84	3·03	2·55	17·0
Tailing	828	0·15	1·24	8·2
Heads	1,000	1·50	15·03	100·0

Comparative Small Scale Tests with Alphabetical, Rex, and Oil Reagents

Three tests were made on the Callow pneumatic testing machine, in which two lots of 1000 grams each, ground to 60 mesh, were used. The machine was operated on a continuous feed, and continuous discharge, as in practice. The concentrate was recleaned on the Ruth testing machine.

Test No. 47:

Reagents used: 6 pounds lime per ton; 1 pound X-Y reagent per ton.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	390	6·45	25·16	80·9
Middling	150	1·90	2·85	9·2
Tailing	1,404	0·10	1·40	4·5
Clean up	56	0·30	1·68	5·4
Heads	2,000	1·55	31·09	100·0

Test No. 48:

Reagents used: Rex No. 12 reagent with soda ash and NH_4OH.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	158	5·55	8·77	28·2
Middling	114	3·82	4·35	14·0
Tailing	1,656	0·97	16·06	51·6
Clean up	72	2·67	1·92	6·2
Heads	2,000	1·56	31·10	100·0

Test No. 49:

Reagents used: Fuel oil, 1 pound per ton; pine oil and 4 pounds of lime per ton.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	258	9·15	23·61	81·1
Middling	52	2·90	1·51	5·2
Tailing	1,581	0·12	1·90	6·5
Clean up	109	1·93	2·10	7·2
Heads	2,000	1·46	29·12	100·0

Large Scale Tests using Rex Flotation Reagents

A number of large scale tests were made, using about one ton of the ore for each test, as follows:—

Test No. 1:

Amount tested: 1,932·5 pounds.
Apparatus used: Ruth four cell flotation machine (standard size).
Degree of crushing: 60 mesh.
Dilution: High.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	36·0	15·10	5·44	18·7
" No. 2	57·0	11·90	6·78	23·3
" No. 3	42 0	10·55	4 43	15·3
" No. 4	54·0	8·45	4·56	15·7
Tailing	1,743·5	0·45	7·85	27·0
Heads	1,932·5	1·55	29·95	100·0

Recovery 73·0 per cent. Ratio of concentration 10·2 to 1.

Test No. 2:

Amount tested: 1,915 pounds.
Apparatus used: Ruth four cell flotation machine (standard size).
Degree of crushing: 80 mesh.
Dilution: High.
Concentrate taken off first three cells: middling off fourth, and returned.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	44·0	17·85	7·85	26·9
" No. 2	46·0	15·70	7·22	24·7
" No. 3	33·0	12·90	4·26	14·6
Tailing	1,792·0	0·55	9·86	33·8
Heads	1,915·0	1·70	32·55	100·0

Recovery 66·2 per cent. Ratio of concentration 15·6 to 1.

Test No. 5:

Amount tested: 1,941·5 pounds.
Apparatus used: Ruth four cell flotation machine (standard size).
Degree of crushing: 80 mesh.
Dilution: High.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	46·5	13·05	6·07	20·9
" No. 2	61·0	11·80	7·20	24·8
" No. 3	39·0	9·25	3·61	12·4
" No. 4	49·0	6·95	3·41	11·2
Tailing	1,746·0	0·50	8·73	30·1
Heads	1,941·5	1·60	31·06	100·0

Recovery 69·9 per cent. Ratio of concentration 9·93 to 1.

Re-run of tailing from test No. 5:

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	0·50	4·50	0·02	0·1
" No. 2	0·75	5·30	0·04	0·1
" No. 3	1·50	7·00	0·10	0·3
" No. 4	1·25	7·20	0·09	0·3
Tailing	1,725·75	0·50	8·65	29·3
Feed	1,729·75	0·50	8·65	30·1

Total recovery 70·7 per cent. Ratio of concentration 9·73 to 1.

Test No. 6:

Amount used: 1,947·5 pounds.
Apparatus used: Ruth four cell flotation machine.
Degree of crushing: 80 mesh.
Dilution: High.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	38·0	16·90	6·42	21·7
" No. 2.	43·0	14·00	6·02	20·4
" No. 3	31·5	11·40	3·59	12·1
" No.	31 0	8·75	2·71	9·2
Tailing	1,804·0	0·60	10·82	36·6
Heads	1,947·5	1·55	30·19	100·0

Recovery 63·4 per cent. Ratio of concentration 13·6 to 1.

Test No. 8:

Amount tested: 1,834·5 pounds.
Apparatus used: Ruth four cell flotation machine.
Degree of crushing: 80 mesh.
Dilution: one of solids to six of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	113·5	10·25	11·63	43·9
" No. 2	96·5	5·55	5·36	20·3
" No. 3	90·0	4·60	4·14	15·6
" No. 4	62·5	4·05	2·53	9·6
Tailing	1,472·0	0·19	2·80	10·6
Heads	1,834·5	1·65	30·27	100·0

Recovery 89·4 per cent. Ratio of concentration 5·1 to 1.

Test No. 9:

Amount tested: 1,750·5 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to six of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	47·5	13·95	6·63	27·4
" No. 2	54·0	9·90	5·35	22·1
" No. 3	41·0	6·75	2·77	11·5
" No. 4	89·5	6·15	5·50	22·7
Tailing	1,518·5	0·26	3·95	16·3
Heads	1,750·5	1·55	27·13	100·0

Recovery 83·7 per cent. Ratio of concentration 7·5 to 1.

Test No. 10:

Amount tested: 1,591 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to five of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	49·5	12·07	5·97	24·2
" No. 2	35·0	14·70	5·15	20·8
" No. 3	43·0	10·38	4·46	18·0
" No. 4	47·0	10·72	5·04	20·4
Tailing	1,416·5	0·29	4·11	16·6
Heads	1,591·0	1·35	21·48	100·0

Recovery 83·4 per cent. Ratio of concentration 9·1 to 1.

Test No. 33:

Amount tested: 1,696 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: High.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	84·0	10·52	8·84	33·1
" No. 2	61·5	9·35	5·75	21·6
" No. 3	54·0	7·27	3·93	14·7
" No. 4	52·0	7·47	3·88	14·5
Middling	130·5	1·37	1·79	6·7
Tailing	1,314·0	0·19	2·50	9·4
Heads	1,696·0	1·60	27·14	100·0

Recovery 83·9 per cent. Ratio of concentration 6·75 to 1.

Test No. 46:

Amount tested: 1,878 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to fourteen of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	53·5	8·62	4·61	17·6
" No. 2	39·5	10·17	4·02	15·4
" No. 3	22·5	10·60	2·38	9·1
" No. 4	59·5	9·10	5·41	20·7
Tailing	1,703·0	0·57	9·71	37·2
Heads	1,878·0	1·65	30·99	100·0

Recovery 62·8 per cent. Ratio of concentration 10·7 to 1.

Large Scale Tests using X-Y Reagents

Two tests were conducted, one on the Ruth machine, and one on the Callow, on about one ton of ore using the Alphabetical reagent, X-Y mixture.

Test No. 18:

Amount tested: 1,652·5 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to six of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate	225·0	6·70	15·07	68·9
Middling	86·5	4·76	4·12	18·8
Tailing	1,341·0	0·20	2·68	12·3
Heads	1,652·5	1·72	28·42	100·0

Recovery 80·2 per cent, assuming 60 per cent recovery of values in middling.
Ratio of concentration 6 to 1.

Test No. 23:

Amount tested: 1,661·5 pounds.
Apparatus used: Callow standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to five of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate	157·5	8·50	13·39	61·4
Middling	91·0	4·90	4·46	20·5
Tailing	1,413·0	0·28	3·96	18·1
Heads	1,661·5	1·49	24·76	100·0

Recovery 73·7 per cent, assuming 60 per cent recovery of values in middling.
Ratio of concentration 8 to 1.

Large Scale Test using Fuel Oil, Pine Oil, and Lime

Test No. 52:

Two tons of ore were ground in the ball mill to the following degree of fineness:—

+100 mesh	16·1 per cent
−100+150	11·9 "
−150+200	34·2 "
−200	37·8 "

TEST MADE ON CALLOW STANDARD MACHINE

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate	217·0	15·75	34·18	71·2
Tailing	3,453·0	0·40	13·81	28·8
Heads	3,770·0	1·35	51·00	100·0

Recovery 71·2 per cent. Ratio of concentration 17·4 to 1.

SUMMARY AND CONCLUSIONS

The experimental work conducted on this ore shows that it is amenable to concentration by flotation methods. A very satisfactory separation of the copper minerals from the iron pyrite and gangue can be obtained, with a good recovery of the copper values, and an economical ratio of concentration.

The small scale tests show that such reagents as oil, with the addition of lime, Rex reagents, or the Alphabetical reagents give good results. However, in the tests run for comparative results under the same conditions and procedure, oil and the Alphabetical reagents give much better results than the Rex reagents.

While the small scale tests give good results with the use of Rex reagents, these reagents have never been applied in practice on a large scale, and until this is done, the feasibility of their use cannot be considered as practicable.

The results from the large scale experimental work are not as satisfactory as those from the smaller scale tests. This is due to the fineness of crushing. In these tests the ore was reduced to 80 mesh, while finer grinding is necessary for better recoveries and grade of concentrates. Since these tests were run a Dorr classifier with bowl attachment has been installed in closed circuit with the ball mill. With this arrangement any degree of crushing can be obtained to 200 mesh, and a number of tests will be conducted on a large scale, and this report supplemented with the results of these tests.

A number of tests will also be made with the use of other reagents, and the results of this work will be contained in the supplement to this report.

PLATE I

Hidden Creek ore, x 100

200 mesh

Hidden Creek ore, x 100
Py = pyrite, Cp = chalcopyrite
Sl = sphalerite, Asp = arsenopyrite

200 mesh

PLATE III

Thin section of Hidden Creek ore, x 100
Black is sulphide

200 mesh

(2)

THE FLIN FLON COMPLEX SULPHIDE ORE

C. S. Parsons and R. J. Traill

Test No. 142

Location.—The Flin Flon ore-body is situated at the southeast end of Flin Flon lake, which drains through a series of lakes into the Saskatchewan river. The ore-body lies in township 67, ranges XXIX and XXX, west, approximate latitude 54° 45′ and approximate longitude 101° 55′—four miles east of the second meridian, west. It lies practically on the boundary of the provinces of Manitoba and Saskatchewan, in the southwest corner formed by the right angle connection of this boundary line. It is 68 miles by air line NNW. from The Pas, on the Canadian National railway, the distributing centre for the district.

Extent of Ore-body.—The ore-body which skirts the southeast shore of the lake strikes with the country rock, and dips approximately 70° east. From the records of the diamond drill holes at the south end of the deposit, it would appear that it pitches at a low angle to the south. Horses of unmineralized greenstone separate it into more or less independent lenses. The deposit has a known length of 2,600 feet, and has been proven to a depth of 900 feet, over a length of 1,000 feet. Its greatest width is 400 feet. From the results of diamond drilling and underground development work, the total tonnage has been calculated to be sixteen million, exclusive of horses of greenstone. This estimate makes no allowance for possible ore below the 900 foot level, or at depth in the line of pitch, at the south end of the ore-body.

Mineralogical Composition.—The chief metallic minerals are pyrite, sphalerite, and chalcopyrite. Galena has been found in vugs in the otherwise unmineralized rock, but does not occur in quantity in the ore-body. Small amounts of pyrrhotite and arsenopyrite, are present, and the analysis of the ore shows the presence of traces of bismuth, which would be present as bismuthinite. Native copper is found in leaf form as a secondary product in the upper sulphide zone. Gold and silver are present. It has been determined that the precious metals are evenly distributed through all the sulphides. This has been demonstrated by the analysis of products from the differential flotation tests on the sulphide ore. The principal gangue is some mineral of the chlorite group.

Characteristics of the Ore-body.—The ore-body consists of three zones and two types of ore—a hanging wall zone of disseminated ore, a centre zone of solid sulphides, and a footwall zone of disseminated ore. In some places the sulphide zone is in direct contact with the hanging wall, but as a rule, a selvage of disseminated ore separates it from the hanging wall. On the footwall there is found invariably a considerable width of disseminated ore. Contact with the walls, and as a general rule between the two types of ore, is well defined. The richest copper ore is found in the disseminated footwall zone, which gives values of from three to five per

cent copper. The sulphide ore carries about 1·65 per cent copper, the average for the whole ore-body being approximately 1·9 per cent. Zinc, which averages about 3·8 per cent for the whole ore-body, shows higher values in the sulphide ore.

Exploration and Development.—The ore-body was discovered in the summer of 1915. During the winter of 1915-16, work was confined to surface sampling and cross trenching. During 1916, 6,000 feet of drilling was done by New York and Boston interests. During 1917 and 1918 forty-four holes were drilled by Toronto interests, representing a total lineal footage of 25,664 feet. In March, 1920, an option was taken by New York and Canadian interests, and since that date two shafts were sunk over 500 feet apart. One shaft reached the 200 foot level where 210 feet of cross-cutting was done to traverse the ore-body, and approximately 100 feet of drift was driven southward. The No. 2 shaft was sunk to the 100-foot level and a cross-cut approximately 165 feet long driven at this level.

Shipments Received for Experimental Test Purposes.—In June, 1920, application was made by the Mining Corporation of Canada, who hold the option on the property, to the Department of Mines for assistance in solving the metallurgical problems in connection with the treatment of the sulphide ore. Later this was extended to cover the disseminated ore. On June 15, 1920, there was received at the Ore Dressing and Metallurgical Laboratories, a shipment of 100 pounds of sulphide ore, and on April 22, 1921, a further shipment of 129 pounds of similar ore was received. Both these shipments were representative, being portions of the drill cores, crushed to about one-half inch, from diamond drilling operations. A shipment of the disseminated ore has been received, the experimental work on this ore will be conducted during 1922.

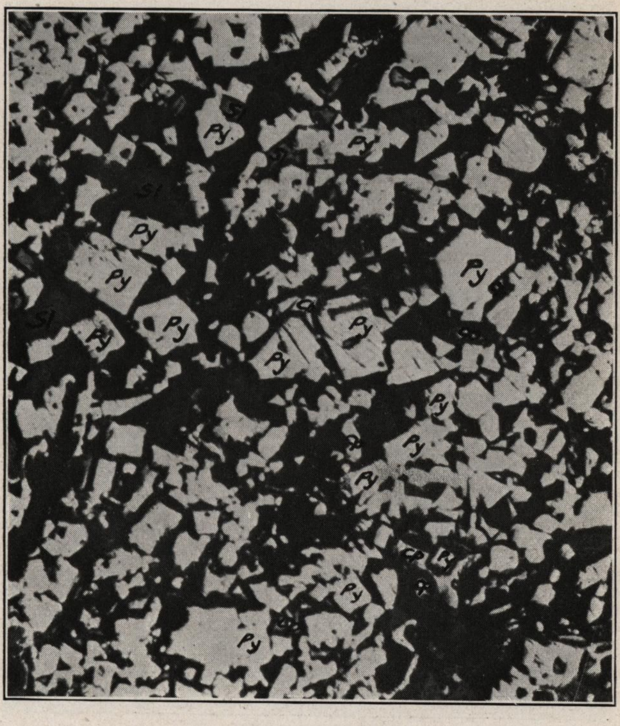
Possible Lines of Investigation for the Metallurgical Treatment of the Sulphide Ore.—On account of the nature of this ore, which consists of practically solid sulphides of iron, zinc, and copper, the gangue minerals being less than ten per cent, it presents a most intricate metallurgical problem to determine the most economical method of treatment. Three of the most promising lines of investigation were decided on as follows:—

1. The concentration of the copper mineral by selective flotation, and the separation by differential flotation of the chalcopyrite, sphalerite, and pyrite, resulting in copper and zinc products, suitable for subsequent reduction to the metals.

2. The hydro-metallurgical treatment of the ore, or of concentration products, by roasting, leaching, and electrical deposition of the metals.

3. Pyritic smelting followed by reduction of the matte in converters to blister copper.

The experimental work conducted in the Ore Dressing and Metallurgical Laboratories was confined to the first line of investigation. A few small laboratory tests were made on the roasting and leaching of the ore, but the laboratories were not equipped to carry out extensive tests from which reliable conclusions could be drawn. The third is a matter for study of conditions at metallurgical plants operating on a similar class of ore, and will require the installation of a small unit at the property to demonstrate the feasibility of this method of treatment.



Flin Flon ore, x 100

☐ 200 mesh

Flin Flon ore, x 100

200 mesh

Flin Flon ore, x 100
Py = pyrite, Cp = chalcopyrite
Sl = sphalerite

200 mesh

PLATE VII

Thin section of massive sulphide ore like that of Plates IV, V, and VI, x 100
Black is sulphide, light is carbonate

200 mesh

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Microscopic Examination of the Sulphide Ore.—A number of polished sections were made and these examined under the microscope to determine the special characteristics of the minerals and their relation to each other. The conclusions arrived at were practically the same as those of Prof. Alfred Wandke, of Cambridge, Mass., who examined three type specimens of the ore. Prof. Wandke's findings were as follows:—

> The gangue is essentially carbonate, either calcite or magnesium, or iron bearing, with very minor amounts of sericite, quartz, and rutile. It is but a small proportion of the ore, and being a carbonate, would not interfere with the fine grinding of the rock. The quartz is sparely developed and moulds about the pyrite, sometimes a little chlorite occurs on the edges of some of the pyrite crystals, and chalcopyrite seems intimate with the chlorite.
>
> The minerals present are pyrite, sphalerite, chalcopyrite, arsenopyrite, a very little bornite, and chalcocite, and a few specks resembling galena. The pyrite occurs both as granular aggregate and as distinct crystals measuring up to ·2 mm. in section. Pyrite probably encloses both sphalerite and chalcopyrite. The chalcopyrite measures frequently under ·02 mm. (less than 400 mesh). The sphalerite cements the pyrite, often occurs in bands, and although usually free from chalcopyrite, it may nevertheless enclose minute ·1 mm. (150 mesh) grains of chalcopyrite. The chalcopyrite is the chief copper mineral, the only one present, for the bornite and chalcocite occur in insignificant amounts. The mineral cements and is intergrown with grains of pyrite and sphalerite. While the grains of chalcopyrite may attain a diameter of ·5 mm. in section, it rarely runs above ·2 mm. (65 mesh) and as a rule the grains measure less than ·1 mm. (150 mesh). Hence very fine grinding (to slime) would be necessary to liberate the chalcopyrite. Sometimes the hand specimens and the polished section show tarnishing, suggesting surface enriching solutions, but the enrichment is always sub-microscopic. The arsenopyrite occurs in clean-cut crystals enclosed by pyrite. It is almost always surrounded by pyrite and makes up but a fraction of one per cent of the ore.

He made a special report on the occurrence of the gold, but found no mineral which would give a clue as to its association. Since it is possible when the gold is coarse to detect the native metal in ore that runs less than $2 per ton, it seems certain that in the Flin Flon ore, the gold is extremely finely divided and so avoids detection. He states that from evidence furnished by other districts, it is suggested that the gold may go with the sphalerite and chalcopyrite. This does not seem to be the case, however, as the results of the differential flotation tests show that the gold is evenly distributed in the ore, occurring with all three minerals, the pyrite, sphalerite, and chalcopyrite.

He also mentions that the extreme fineness of the grains of chalcopyrite have indicated to him that the flotation process might not be advisable. However, the results of the flotation tests show that on grinding to 200 mesh, to which fineness the ore reduces readily, a high grade copper concentrate is obtained, with good recoveries, and a zinc product is made, with a recovery of from 55 per cent to 60 per cent of the zinc. As far as the copper mineral is concerned, the ore is amenable to the flotation process.

It was also determined that the gangue mineral was principally carbonate, and that magnesia was greatly in excess of calcium. It is possible that the mineral which Prof. Wandke takes for galena, is bismuthinite, as bismuth has been detected in the electrolytic deposition of the copper.

The ore has been found to contain 0·5 per cent arsenic. This is roughly equivalent to 1 per cent arsenical pyrites.

Tests on the Concentration and Separation of the Minerals.—Small shipments of the ore were sent by the Mining Corporation of Canada to the General Engineering Company, Salt Lake City, Utah, to the company's Cobalt works, and to the Ore Dressing and Metallurgical Laboratories, Mines Branch, Department of Mines, for experimental work on the concentration and separation of the minerals.

The results of the tests conducted by the General Engineering Company are summarized below:—

Test No. 3:

Amount tested: 1,000 grams.
Apparatus used: Callow testing machine.
Degree of crushing: 200 mesh.
Object of test: Concentration of the copper values.
Reagents used: 11 pounds lime per ton; 1 pound X-Y mixture per ton.

Products	Analysis						Recov. Per cent	Conc. Ratio
	Cu. per cent	Zn. per cent	Fe. per cent	Insol. per cent	Au.-oz.	Ag.-oz.		
Copper concentrate	13·70	5·3	33·0	5·2	.....	.....	86·6	10·6 : 1
Tailing	0·22	5·4	.....	.....	0·06	1·00	.....	.....

Test No. 4:

Amount tested: 1,000 grams.
Apparatus used: Callow testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals, producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used: 14 pounds lime per ton; 0·1 pound X-cake; 4 pounds Wattsons 1-B (equivalent to 0·6 pound oil); producing a copper concentrate.

Refloated, using 6 pounds H_2SO_4; 6·5 pounds residue Wattsons 1-B; 1 pound G. E. Co. No. 209; 0·1 pound G. E. Co. No. 56, producing a zinc concentrate.

Products	Analysis						Recov. Per cent	Conc. Ratio
	Cu. per cent	Zn. per cent	Fe. per cent	Insol. per cent	Au.-oz.	Ag.-oz.		
Copper concentrate	8·00	5·6	33·8	5·4	0·26	4·50	83·1	6·2 : 1
Zinc concentrate	1·25	37·6	16·7	.....	0·08	2·12	60·3	1·3 : 1
Tailing	0·22	1·3	.....	.....	0·07	0·90	.....	.....

Test No. 14:

Amount tested: 1,000 grams.
Apparatus used: Callow testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals, producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used: 14 pounds lime; 0·15 pound of X-cake; 6 pounds Wattsons 1-B (equivalent to 0·9 pound oil), producing a copper concentrate.
Refloated, using 1 pound of G. E. Co.'s No. 209, 0·75 pound Wattsons residue 1-B; 0·10 pound G. E. Co.'s No. 56, producing a zinc concentrate.

Products	Analysis						Recov. per cent	Conc. ratio
	Cu. per cent	Zn. per cent	Fe. per cent	Insol. per cent	Au.-oz.	Ag.-oz.		
Copper concentrate	6·70	7·8	32·8	5·6	0·22	3·90	83·4	5·3 : 1
Zinc concentrate	1·20	38·5	18·2	2·0	0·08	1·75	54·8	18·3 : 1
Tailing	0·25	0·24	.....	.....	0·07	0·80	.....	.....

The results of the tests conducted by the Ore Dressing and Metallurgical Division of the Mines Branch, Department of Mines, are summarized below—

Test No. 13:

Amount tested: 1,000 pounds.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals, producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used:
Flotation of copper: 40 pounds lime per ton added to ball mill. 4 pounds X-Y per ton added to cells.
Flotation of zinc: 14-B wood tar and Wattsons residue. This was unsuccessful.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc	86	13·0	4·8	2·02		41·4	6·5	.....	.....
Copper midd	152	6·3	5·0	0·12	1·72	35·5	12·0	.....	.....
Tailing	779	0·8	6·6	0·08	0·74	23·1	81·5	89·0	51·0

Test No. 14:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used:
Flotation of copper: 40 pounds per ton lime added to ball mill; 5 pounds per ton X-Y reagent added to cell.
Flotation of zinc: 10 pounds per ton of soda ash.
1 pound per ton No. 34 gravity fuel oil.

Products	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc.....	146	14·2	4·4	0·10	1·38	78·5	10·2	.....	.....
" midd.....	159	2·3	5·9	0·14	1·27	13·8	14·9	.....	.....
Zinc conc.....	96	0·9	29·0	0·16	0·10	3·3	44·1	.....	.....
" midd.....	65	0·5	6·3	0·10	1·10	1·2	6·5	.....	.....
Tailing.....	568	0·15	2·7	.....	.....	3·2	24·3	.....	.....

Test No. 15:

Amount used: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals, producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used:
Flotation of copper: 40 pounds per ton lime added to ball mill; 5 pounds per ton X-Y reagent added to cell.
Flotation of zinc: Dewatered copper tailing; 8 pounds per ton soda ash to cell; 1 pound per ton No. 34 gravity fuel oil.

Product	Weight grams	Analysis			Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au. and Ag.-ozs.	Cu.	Zn.	Au.	Ag.
Copper conc.....	161	12·5	2·7	1·54	77·9	7·7	.....	.....
" midd.....	110	1·4	7·0	0·90	6·0	13·6	.....	.....
Zinc conc.....	90	2·4	36·5	1·70	8·4	58·3	.....	.....
" midd.....	100	0·8	5·5	0·40	3·1	9·8	.....	.....
Tailing.....	542	0·22	1·1	0·71	4·6	10·6	.....	.....

Test No. 16:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used:
Flotation of copper: 24 pounds per ton lime added to ball mill; 4 pounds per ton X-Y reagent to cell.
Flotation of zinc: Dewatered copper tailing: 1 pound per ton P.T.T. Co. No. 1580 to cell. This was unsuccessful.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc.....	106	17·8	3·6	0·10	2·14	69·2	6·0	15·8	21·4
Copper midd.....	148	3·6	7·2	trace	2·00	19·6	16·8	4·2	28·0
Tailing.....	766	0·4	6·4	0·07	0·70	11·2	77·2	80·0	50·6

Test No. 18:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Concentration of the copper mineral.
Reagents used: 30 pounds per ton lime to ball mill; 5 pounds per ton X-Y reagent to cell.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Fe. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc.....	158	14·9	28·7	0·12	.....	86·0	.....	23·7	.....

Test No. 19:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used: Flotation of copper:
24 pounds per ton lime added to ball mill; 5 pounds per ton X-Y reagent to cell.
Flotation of zinc:
Dewatered copper tailing.
8 pounds per ton soda ash added to cell.
3 pounds per ton Wattsons residue 1-B.
1 pound per ton copper sulphate.
1 pound per ton No. 34 gravity fuel oil.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.....	145	16·7	4·4	24·3	0·10	87·94	10·56	.....	18·2
" midd.....	29	1·8	8·7	28·3	0·10	1·89	4·17	.....	3·6
Zinc conc.....	78	0·45	40·7	17·9	0·11	1·27	52·52	.....	10·8
" midd.....	74	0·5	8·9	33·7	0·07	1·34	10·90	.....	6·6
Tailing.....	695	0·3	1·9	35·4	0·07	7·56	21·84	.....	60·8

Test No. 20:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Concentration of the copper mineral.
Reagents used: 20 pounds per ton lime added to ball mill.
3 pounds per ton X-Y mixture added to cell.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.....	93	20·8	.....	.....	0·13	70·4	.....	.....	.....
Copper midd.....	39	7·4	.....	.....	.....	10·6	.....	.....	.....

Test No. 21:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used:
Flotation of copper:
16 pounds per ton lime added to ball mill.
0·3 pound per ton X-Y reagent added to cell.
Flotation of zinc:
Dewatered copper tailing.
8 pounds soda ash per ton.
1 pound fuel oil per ton.
Flotation of zinc was not successful.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc.....	138	17·4	3·2	0·08	2·64	87·4	7·37	13·8	.....

Test No. 22:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used:
Flotation of copper:
20 pounds per ton lime added to ball mill.
0·3 pound per ton X-Y reagent to cells.
Flotation of zinc:
Dewatered copper tailing.
5 pounds per ton lime.
8 pounds per ton soda ash.
1 pound per ton fuel oil and pine oil.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc.....	129	15·7	2·7	0·12	2·58	73·7	.....	.....	.....
" midd.....	46	2·6	6·8	0·12	1·84	.....	.....	.....	.....
Zinc conc.....	61	.....	44·5	.....	.....	.....	45·2	.....	.....
" midd.....	116	0·58	6·7	0·08	1·18	.....	.....	.....	.....
Tailing.....	.....	.....	2·8	.....	.....	.....	.....	.....	.....

Test No. 23:

Same procedure as in test No. 22.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.....	130	18·0	3·7	26·4	0·16	85·0	.....	.....	.....
Zinc conc.....	90	.....	39·5	15·4	0·18	.....	59·0	.....	.....
Tailing.....	594	0·2	0·8	36·6	0·11	.....	.....	.....	.....

Test No. 24:

Same procedure as in tests Nos. 22 and 23.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.....	222	10·9	4·5	27·7	0·10	88·0	17·0	.....	.....
Zinc conc.....	61	.....	42·5	14·2	0·20	.....	44·0	.....	.....
" midd.....	150	0·6	8·6	31·7	0·14	.....	21·9	.....	.....
Tailing.....	590	0·1	1·7	37·1	0·06	.....	17·0	.....	.....

Test No. 32:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used:

Flotation of copper:

20 pounds lime per ton to ball mill.
0·3 pound X-Y reagent to cells.

Flotation of zinc:

Dewatered copper tailing.
10 pounds per ton caustic soda.
8 pounds per ton lime.
1 pound per ton fuel oil.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.....	200	8·80	2·4	.....	.....	67·8	7·9	.....	.....
" midd.....	96	2·35	4·3	.....	.....	8·7	6·8	.....	.....
Zinc conc.....	100	3·42	26·3	.....	.....	13·2	43·5	.....	.....
" midd.....	140	0·70	7·3	.....	.....	3·8	16·9	.....	.....
Tailing.....	486	0·35	3·1	.....	.....	6·5	24·8	.....	.....

Test No. 34:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Preferential flotation of the minerals.
Reagents used:
Flotation of copper:
40 pounds per ton lime added to ball mill.
0·2 pound X-Y reagent added to cells.
Flotation of zinc:
6 pounds per ton soda ash.
1 pound per ton gravity fuel oil.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.....	194	10·95	3·6	29·7	0·10	65·5	11·6	.....	.....
" midd.....	89	7·75	4·9	31·6	0·08	21·2	7·2	.....	.....
Zinc conc.....	93	2·30	35·0	21·2	0·15	6·6	54·1	.....	.....
" midd.....	92	1·05	7·8	34·1	0·06	3·0	11·9	.....	.....
Tailing.....	608	0·20	1·5	.....	.....	3·7	15·2	.....	.....

Cyanidation of Flotation Tailing.—The tailing from test No. 34 was agitated in cyanide solution for the recovery of the gold and silver. The tailing after being treated in this manner gave traces of gold and silver, showing that the precious metal values could be recovered in this way. However, the cost of treatment would very probably prohibit the use of this method.

Microscopic Examination of Flotation Products.—Samples of flotation products from tests Nos. 15, 19, and 24, were sent to the Mining Corporation of Canada, who submitted same to Prof. Alfred Wandke, for microscopic examination. A copy of his discussion and conclusions is given below:—

Discussion of Results.—The microscopic examination of the flotation products of Flin Flon ore was made on the following types of products:—

Copper concentrates, copper middlings, zinc concentrates, zinc middlings, and tails.

The examination was made with the following objectives in view:—

1. To study the degree of liberation of the various minerals of these products after being ground and floated.
2. To determine whether even finer grinding would be necessary to further liberate the various minerals, and so effect an even higher concentration of both copper and zinc.
3. To see whether any mineral was present in the various products that might account for the gold and silver content of the ore, and whether finer grinding might aid in recovering the precious metals.

Degree of Liberation.—The examination showed that each of the three chief sulphide minerals of the ore—pyrite, chalcopyrite and sphalerite—occurs for all purposes practically free from any other sulphide or gangue minerals. In examining the copper concentrates fully 90 to 95 per cent of the partieles present occur uncontaminated. The remaining 5 to 10 per cent occur intergrown with either pyrite and sphalerite in such fine division (minus 400 mesh) that further grinding would be impracticable.

The zinc concentrates showed the sphalerite to be practically entirely liberated, the copper and iron present being largely accounted for by grains of chalcopyrite and pyrite.

In the middlings from both the copper and zinc concentrating experiments the contaminating sulphides, whether chalcopyrite, sphalerite, or pyrite are present in free grains. The middling products thus show an imperfect separation. Regrinding of the middlings would be unnecessary to cause further liberation of the chalcopyrite or sphalerite as the case may be.

In the tails the chalcopyrite or sphalerite are both present largely as well liberated grains, only a minor portion of either chalcopyrite or sphalerite occurring locked up in the pyrite. Further grinding, it seems, would hardly liberate this locked up material for it rarely equals 400 mesh in size.

Would Finer Grinding Help the Degree of Liberation.—As stated above, the various sulphide minerals occur remarkably well liberated in the various products. Much of the material already ranges well below minus 200 mesh in size, and it would hardly seem feasible to even consider finer grinding because of the prohibitive cost.

Occurrence of the Gold and Silver.—Each of the products was scrutinized with great care to see if any clue could be obtained as to the occurrence of the gold and silver. No mineral was detected that would indicate the presence of gold and silver. It is concluded, therefore, that these precious minerals occur in a state of extreme subdivision and will probably defy concentration. They apparently occur associated with either the chalcopyrite, the sphalerite, or the pyrite. Thus the microscopic examination substantiates what the assays had already indicated regarding the occurrence of gold and silver in these ores.

CONCLUSIONS

1. The study of ore specimens had already indicated that this Flin Flon ore would have to be very finely ground in order to selectively concentrate either the chalcopyrite or the sphalerite. The flotation products show that the requisite degree of fine grinding was carried out for but a very minor amount of composite grains are present.

2. In the ore specimens no clue was found which would suggest the possible occurrence of the gold and silver. In the products the gold and silver still defy detection.

3. Since the gold and silver form necessary recoverable values in this ore, it seems hardly likely that any scheme of preferential flotation will permit the recovery of these precious metals. The reasons for this statement are as follows:—

(*a*) As judged from their appearance under the microscope the various sulphide grains seem to be at least 60 per cent through 200 mesh in size. This crushing has liberated the various sulphides to a remarkable degree. The composite grains show the enclosed sulphide particles to be well under 200 mesh in size, and hence of a size that wil defy liberation by grinding.

(*b*) The assays seem to indicate that the gold and silver occur more or less indifferently with either the chalcopyrite, the sphalerite, or the pyrite. Hence flotation merely separates the chief sulphide minerals of the ore and leaves the gold and silver almost equally disseminated in each.

4. It would appear, therefore, that the treatment of this ore must be accomplished either by direct smelting or by a roasting and leaching process.

PLATE VIII

Copper concentrate from Flin Flon sulphide ore, x 100

200 mesh

PLATE IX

Copper middling from Flin Flon sulphide ore, x 100

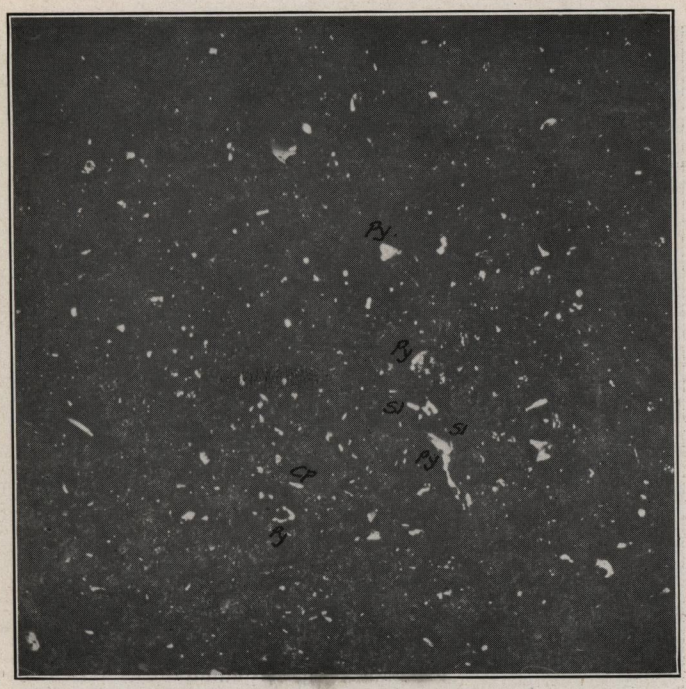
200 mesh

PLATE X

Zinc middling from Flin Flon sulphide ore, x 100

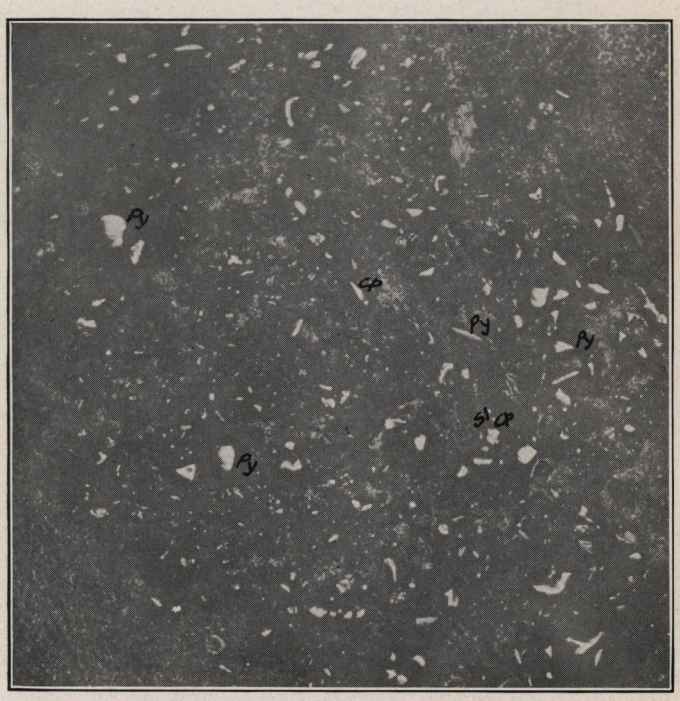
200 mesh

PLATE XI



Zinc concentrate from Flin Flon sulphide ore, x 100

200 mesh



Tailing from Flin Flon sulphide ore, x 100
Py = pyrite, Cp = chalcopyrite, Sl = sphalerite

200 mesh

Roasting and Leaching Tests

Roast No. 1

Roasting.—A sample of 100 grams of ore, 200 mesh, was roasted in an electric muffle furnace (multiple unit type) under the following conditions and with the following results:—

The 100 grams was spread over an area of 8″ × 3″, doors were kept closed during roast, except when charge was hand rabbled, and only inlet for air was $\frac{1}{8}$″ circular hole in front door, and possibly to a slight extent, space around the door.

A suction was applied to rear door to draw off excess SO_2 gas, and also to create a slight in-draught of air to assist in slow oxidation of sulphur.

Temperature observations:—

10 a.m. (15 mins. after placing charge)	300° C.
10.30 a.m.	350° C.
11.00 a.m.	400° C.
11.30 a.m.	440° C.
12.00 noon	450° C.

This temperature maintained until 3 p.m., and then increased gradually until 4.30 p.m., when temperature reached 650° C.

Charge withdrawn at 4.30 p.m.
Time of roast, 6½ hours.
Charge showed a loss in weight of 16·5 per cent.
Temperature range, minimum 300°, maximum 650° C.
Rabbling was performed every half hour.

Leaching.—Two gram samples were leached at a point below boiling temperature for 1 hour. Head samples run as in standard analysis:

Results of coarse material:

	%	%
Head sample Cu	2·35	
Water soluble Cu	0·58—extraction	24·7
H_2SO_4 soluble Cu	2·19—extraction	93·2
Head sample Zn	6·75	
Water soluble Zn	4·36—extraction	64·6
H_2SO_4 soluble Zn	6·12—extraction	90·7

Insoluble residue from 4 p.c. acid leach, 65%
Heads SO_3 in sample, 10·50%

Dry ground, about 200 mesh:

	%	%
Water soluble Cu	none	
H_2SO_4 soluble Cu	2·10—extraction	89·3
Water soluble Zn	4·10—extraction	60·7
H_2SO_4 soluble Zn	6·4 —extraction	94·8

Insoluble residue from 4% acid leach, 59·60%—Fe. 55·25%
Sulphide S, 0·4%—Total S. 4·60%

Roast No. 2

Roasting.—Sample of 100 grams 20 mesh ore roasted in E. M. furnace under following conditions and with the following results:—

General conditions similar to test No. 1, with exception that front door of furnace was left open slightly ($\frac{1}{2}''$) to allow a current of warm air to pass over material. Suction applied at rear end of furnace to an extent that no sulphur fumes could be detected coming out of front door. Rabbled every half hour.

Temperature observations:—

Starting temperature, 9.30 a.m	450° C.
10.00 a.m	475
10.30 a.m	480
11.00 a.m	500
11.30 a.m	550
12.00 noon	585
12.30 p.m	610

Roast withdrawn at 12.30. Time of roast, 3 hours.
Temperature range, min. 450°, max. 610° C.
Charge showed a loss in weight of 17·5 per cent.
Leaching.—Method same as in test No. 1.

Results on coarse material:

	%		%
Heads—Cu	2·35		
Water soluble Cu	0·36	—extraction	15·3
4% H_2SO_4 soluble Cu	2·18	—extraction	93·0
Heads—Zn	6·77		
Water soluble Zn	3·95	—extraction	58·34
4% H_2SO_4 soluble Zn	5·50	—extraction	81·24
Total S	4·94		
Sulphide S	1·15		
SO_3 in sample	9·50		

Fine dry ground:

4% H_2SO_4 soluble Cu	2·15	—extraction	91·5
4% H_2SO_4 soluble Zn	5·20	—extraction	76·8

A sample drawn at 11 a.m. gave the following analysis on leaching:—

	%		%
Water soluble Cu	none		
4% H_2SO_4 soluble Cu	1·25	—extraction	53·60

	%		%
Water soluble Zn	none		
4% H_2SO_4 soluble Zn	2·65	—extraction	39·14

SUMMARY AND CONCLUSIONS

1. Concentration of the Ore.—The results of the experimental work show no difficulty in the concentration of the chalcopyrite by preferential flotation after grinding to 200 mesh. A copper concentrate is obtained as high as 15 per cent copper with a recovery of 85 per cent of the copper

(3)

THE CONCENTRATION OF GRAPHITE FROM THE NORTH AMERICAN MINE

C. S. Parsons

Considerable experimental work has been conducted in the Ore Dressing and Metallurgical Laboratories on graphite ores from different points in the Dominion. The introduction of the oil flotation process has made radical changes in the treatment of graphite ores. The old methods have been for the most part abandoned, for this newer process simplifies operations, with the production of higher grade products, and with a much higher recovery of the graphite in the ores.

In the Summary Report of the Mines Branch for 1919, under Test No. 126 of the Ore Dressing and Metallurgical Laboratories, a description is given of the experimental work conducted on a large shipment from the Quebec Graphite Company, Buckingham, Que. In this case a process was worked out which was later put into practice by the installation of a new mill by the company. This mill was very successful in treating their ore following the methods outlined by the Department. The test work conducted on graphite ores can be found under Tests Nos. 85, 102, and 103 of the Summary Report for 1918, Tests Nos. 113 and 126 of the Summary Report for 1919, and Tests Nos. 129 and 139 of the Summary Report for 1920. This work has been largely instrumental in the introduction of more advanced methods of treating graphite ores.

Test No. 143

A 41·5 pound sample of graphite ore was received January 3, 1921, at the Ore Dressing and Metallurgical Laboratories from Mr. H. P. H. Brumell, of Buckingham, Quebec. The ore was from the North American Graphite mine, Buckingham, and consisted of graphite flake of a fair size in a gangue mainly of calcite and pyroxene, with small amounts of mica and iron pyrites.

A few specimens were selected from the sample and the balance was crushed down gradually, with frequent screening, in a small jaw crusher and a small set of rolls, until it all passed through 10 mesh. A head sample was cut out of this −10 mesh material, which gave upon analysis, 13·40 per cent carbon.

Concentration tests were desired to determine the possibility of recovering a high percentage of the contained carbon in the form of a high grade coarse flake. Tests were conducted, using flotation, and in some cases concentration on tables was used in conjunction with flotation.

Test No. 1.—Five hundred grams of ore, −10 mesh, were floated in a small Ruth flotation machine, making a concentrate and tailing. The tailing was dewatered, ground 20 minutes in a small pebble jar, and floated again in the Ruth machine. The concentrate from the first flotation was tabled on a small Wilfley table, making a concentrate and a tailing. The following table shows the products and the results obtained:

Product	Weight grams	Per cent C.	Grams C.	Per cent C. values
Table conc.	130	43·65	56·74	86·9
Table tails	69	8·20	5·66	8·6
Flot. conc.	20	9·40	1·88	2·9
Flot. tails	275	0·38	1·04	1·6
	494		65·32	
Heads	500	13·40	67·00	100·0

Test No. 2.—Five hundred grams of ore, −14 mesh were floated in a small Ruth flotation machine. The concentrate from this flotation was tabled, making a concentrate and a tailing. The table and flotation tailings were mixed, dewatered, ground in a pebble jar for 20 minutes, and floated in a Ruth machine. The concentrate from this operation was tabled. The following table shows the products and results obtained:—

Product	Weight grams	Per cent C.	Grams C.	Per cent C. values
1st table conc.	126	45·70	57·58	85·8
2nd table conc.	27	29·05	7·84	11·7
Table tails	41	2·60	1·07	1·6
Flot. tails	284	0·22	0·62	0·9
	478		67·11	
Heads	500	13·40	67·00	100·0

Test No. 3.—Five hundred grams of ore, −20 mesh, were treated in the same way as the ore used in test No. 2. The following table shows the products and results obtained:—

Product	Weight grams	Per cent C.	Grams C.	Per cent C. values
1st table conc.	143	43·25	61·85	88·5
2nd table conc.	20	30·25	6·05	8·7
Table tails	22	2·86	0·63	0·9
Flot. tails	293	0·46	1·35	1·9
	478		69·88	
Heads	500	13·40	67·00	100·0

Conclusions derived from Tests Nos. 1, 2 and 3.—The results of these tests show that while recoveries are good, and the tailings low in carbon content, the grade of concentrates produced is very low, due no doubt to the fact that the graphite flake is not entirely freed from the gangue material. In order to raise the grade, further grinding is necessary, and test No. 4 was conducted along this line.

Test No. 4.—One thousand grams of ore, −10 mesh, were floated in a small Ruth flotation machine, the tailing was ground in a pebble jar for 20 minutes and floated again. The concentrates from both operations were mixed and ground in a pebble jar for 20 minutes, and then floated. The flotation concentrate was tabled on a small Wilfley table, making a concentrate and a tailing. The table concentrate was dried and screened on 80 and 150 mesh screens (Tyler). The following table shows the results obtained:—

Product	Weight grams	Per cent C.	Grams C.	Per cent C. values
Table conc. +80	140	73·40	102·76	73·5
Table conc. +150	40	39·80	15·92	11·4
Table conc. −150	24	44·05	10·57	7·5
Table tails	55	11·00	6·05	4·3
Flot. midds	181	1·90	3·44	2·5
Flot. tails	547	0·20	1·09	0·8
	987		139·83	
Heads	1,000	13·40	134·00	100·0

Conclusions derived from Test No. 4.—A high percentage of carbon values is recovered in the form of coarse flake, but the grade is not as high as could be desired. More regrinding is necessary. A further test was made, the details of which are given under test No. 5.

Test No. 5.—One thousand grams of the ore, which had previously been crushed to 10 mesh, were ground in a small ball mill until practically all the gangue would pass a 20 mesh screen.

Flotation was employed for the separation of the graphite from the gangue. The oils used were kerosene, which was added to the ball mill, equivalent in amount to one pound per ton of ore, and pine oil, which was added to the flotation machines as required to form a desirable froth. Further pine oil and coal oil was added from time to time in recleaning the concentrates. The following method of procedure was carried out:—

The ore was floated, making a rougher concentrate and low tailing. The rougher concentrate was dewatered and ground in a pebble mill, refloated, making a second concentrate and tailing. The second concentrate was dewatered and reground in a pebble mill, refloated, making a cleaner concentrate and tailing. The cleaner concentrate was screened on 65 and 150 mesh. All tailing products and the final concentrate were dried, weighed, sampled, and analysed. The results of this test are given in the following tables:—

Products	Weight grams.	Analysis per cent C.	Content grams C.	Percentage of C. values
Cleaner concentrate	145	87·90	127·46	92·7
Rougher tailing	674	0·35	2·36	1·7
Second tailing	120	2·53	3·04	2·2
Cleaner tailing	49	9·54	4·67	3·4
Heads	1,000	13·75	137·50	100·0

NOTE.—The second tailing could be discarded or returned to the circuit. The cleaner tailing would be returned to the circuit, or treated separately, and a percentage of the carbon values recovered.

SCREENING OF CLEANER CONCENTRATE

Products	Weight grams	Percentage of weight	Analysis per cent C.	Content grams C.	Percentage of C. values
+ 65 mesh	68	46·9	94·40	64·19	46·7
−65 +150 mesh	41	28·3	90·25	37·00	26·9
−150 mesh	36	24·8	73·10	26·32	19·1
Totals	145	100·0	87·90	127·46	92·7

Summary of Results.—One ton of ore contains and yields the following products:—

Content: 2,000 pounds at 13·75% C. = 275 pounds of carbon.
Products: 290 pounds of concentrates at 87·9% C.
of which 136 pounds is +65 mesh material at 94·40% C.
82 pounds is −65 +150 mesh material at 90·25% C.
and 72 pounds is −150 mesh material at 73·10% C.

CONCLUSIONS

The above test shows that a high grade graphite concentrate can be obtained without difficulty with a high recovery of the contained carbon values; that the screened products are of excellent grade; that a good percentage is recovered as coarse flake; that in order to obtain these results regrinding is essential to free the flake from attached particles of gangue.

These results have been obtained from a small test under the most favourable conditions. Whether similar results can be obtained in practice is problematical. The grinding action in large mills may have the tendency to destroy the coarseness of the flake much more than in the treatment of the small lot of this test.

(4)

THE CONCENTRATION AND SEPARATION OF THE MINERALS IN THE LEAD-ZINC-SILVER ORES OF KIMBERLEY, B.C.

C. S. Parsons and R. J. Traill

Methods and processes for the concentration and separation of the minerals in the ores of the Kimberley district, B.C., have been the subject of investigation and research for a number of years by the Consolidated Mining and Smelting Company of Canada, Trail, B.C.

Improvements have steadily advanced to a stage where the difficult ores of the Sullivan mine can be successfully treated by differential flotation, resulting in marked recoveries of the lead and zinc in these ores.

The following tests were made on ores from the North Star mine, on the carbonate ore from the upper portion of the ore-body, and on the sulphide ore found at greater depth. The carbonate ore consisted chiefly of lead carbonate, coated and locked up in a mass of iron oxides, formed by the oxidation of iron sulphides. The sulphide ore was an intimate mixture of lead, zinc, and iron sulphides, the gangue minerals representing less than 10 per cent by weight of the ore.

Test No. 144

A shipment of 370 pounds of lead carbonate ore was received at the Ore Dressing and Metallurgical Laboratories, January 26, 1921, from the North Star mine, Kimberley, B.C.

The ore consisted chiefly of lead carbonate, coated and locked up in a mass of iron oxides, formed by the oxidization of the iron sulphides, and gave an analysis as follows:—

Lead (Pb)	10·85 %	
Iron (Fe)	14·30 "	Fe_2O_3 20·45%
Alumina (Al_2O_3)	4·55 "	
Silver (Ag)	6·55 oz. per ton	
Moisture (H_2O)	3·60%	

The object of the experimental work was to determine a method of concentration for the ore. A number of small scale tests were made by sulphidizing the lead carbonates and oxides for flotation, using gravity concentration on tables, and by the chloride volatilization process.

A series of tests were first conducted using a combination of flotation and table concentration. In these tests the table tailings were sulphidized with sodium sulphide before flotation. It was thought that a long contact with the sodium sulphide was necessary, but later it was found that much better results could be obtained by allowing the ore only a short contact with the sulphidizing reagent before flotation. The results of two of the first series of tests are given below:—

Test No. 4.—One thousand grams of ore crushed to 60 mesh was agitated for 30 minutes with 20 pounds per ton of sodium sulphide salt, $Na_2S.\ 9H_2O$. It was floated and the flotation tailing tabled. The following other reagents were added:—

Soda ash	1	pound per ton
Coal tar creosote	4	"
Pine oil (No. 5 G.N.S)	0·4	"

Product	Weight grams	Pb. per cent	Pb. grams	Recovery of Pb.	Remarks
Flotation conc.	149	31·35	46·71	45·4	74·4 per cent recovery assuming that 70 per cent of the lead in the middling would report in the concentrate.
" midd.	191	7·80	14·90	14·4	
Table conc.	44	44·27	19·48	19·0	
" tailing	360	3·15	11·34	11·0	
" slime	179	5·86	10·49	10·2	

Test No. 9.—One thousand grams of ore crushed to 100 mesh was agitated $1\frac{1}{4}$ hours with 10 pounds per ton of sodium sulphide salt $Na_2S.\ 9H_2O$. It was floated and the flotation tailing tabled:—

Product	Weight grams	Pb. per cent	Pb. grams.	Recovery of Pb.	Remarks
Flotation conc.	155	40·35	62·5	57·6	79 per cent recovery assuming that 70 per cent of the values in the middling would report in the concentrate.
" midd.	171	7·25	12·4	11·4	
Table conc.	43	34·33	14·8	13·6	
" tailing	631	2·98	18·8	17·4	

A second series of tests were conducted, varying the time of contact with the sulphidizing reagent, and the strength of this reagent used. The strength of solution used is given in terms of Na_2S. (Note.—The commercial salt contains varying quantities of water.) The results of these tests, and reagents used, are given in the tables following.

The procedure followed was to grind the ore wet in a ball mill to 200 mesh, dewater, and pass through a filter press. The solution of sodium sulphide made up to strength required was then added to the cake, and the two agitated together for a certain length of time. It was found that the sulphidization of the lead particles, which were free and exposed to the action of the reagent, took place almost instantly, and were easily recovered by flotation. The recovery of the silver values was much more difficult, a maximum of 40 per cent was the best that could be obtained.

In test No. 30 a strong solution of sodium sulphide was left in contact with the ore for 15 minutes. The solution on being tested showed no free Na_2S. The recovery of the silver values was no greater than when the weaker solution was used for shorter periods of contact. It was also found that the lowest tailing carried 3 per cent lead. The difficulty in obtaining high recoveries of the lead and silver values seems to be due to the physical characteristics of the ore. The particles of lead carbonate and the silver values, in whatever form they occur, seem to be locked up in a mass of iron oxides, which completely film them. Wet grinding to 200 mesh is necessary to remove as much of this coating as possible, and even at this fine state of division the minerals are not entirely freed, but remain enclosed in the film of iron oxide. This was demonstrated by taking 1,000 grams of the ore, grinding and tabling successively by stages until a fineness of 200 mesh was reached. It was found at this stage of crushing that no more lead was freed to report as concentrate on the table, and the tailing at this stage gave an assay of 3·5 per cent lead.

A recovery of between 70 per cent and 80 per cent of the lead values, and 40 per cent of the silver values, seems to be the maximum that could be expected by the combined method of sulphidizing for flotation and tabling the flotation tailing. The control of the sulphidizing condition seems to be simple, gives uniform results, and the above recoveries should be obtained quite readily from the ore as submitted for test purposes.

Chloride Volatilization Tests

Two small tests were made to determine whether this process was applicable to the ore. The results obtained, as to recovery of the lead and silver values, were practically the same as those obtained by flotation and tabling.

Test No. 1:

Assay of ore	Silver,	6·55 oz.	Lead,	10·85 %
Assay of tailing	Silver,	4·61 oz.	Lead,	3·41 %
Volatilized	Silver,	35·30 %	Lead,	70·4 %

Test No. 2:

Assay of ore	Silver,	6·55 oz.	Lead,	10·85 %
Assay of tailing	Silver,	5·06 oz.	Lead,	4·80 %
Volatilized	Silver,	29·00 %	Lead,	58·70 %

Reagents Used

Test No.	Strength of solution in dry Na_2S salt	Time	Oils used	Remarks
25	2·20 grams in 500 c. c.	15 mins.	10% C.T.; 80% C.T.C.; 10% pine oil—1 c.c.	Too much frothing oil.
26	2·20 " 1,000 "	15 "	Same as test No. 25—1 c.c.	
27	4·40 " 2,000 "	15 "	10% 1-B; 80% C.T.C.; 10% C.T.; and enough crude turps—1 c.c.	
28	2·20 " 2,000 "	15 "	Same as test No. 27—1 c.c.	
29	4·40 " 4,000 "	15 "	Same as test No. 28—1·5 c.c.	
30	8·80 " 3,000 "	15 "	Same as test No. 29	
31	2·20 " 3,000 "	3 "	Same as test No. 29	Added solution directly in cells.
32	2·20 " 3,000 "	3 "	10% B-14; 20% C.T.; 70% C.T.C.	Solution added directly to cells. Very high grade looking froth.
33	4·40 " 3,000 "	5 "	Same as test No. 32	Na_2S solution added directly in cells. Very high grade looking froth.
34	4·40 " 3,000 "	5 "	X-Y reagent—0·25 c.c.	Na_2S solution added directly in cells. Very watery looking froth and voluminous.
35	4·40 " 3,000 "	5 "	10% B-14; 20% C.T.; 70% C.T.C., plus crude turpentine.	Quite a heavy good froth. This sample was only crushed for 15 minutes.
36	4·40 " 3,000 "	5 "	10% B-14; 20% C.T.; 70% C.T.C—1 c.c.	This test ground very fine for two hours. Fine looking black froth.

Results of Tests

Test No.	Concentrate						Middling						Tailing					
	Weight grams	Assay		Content		% of Pb. values	Weight grams	Assay		Content		% of Pb. values	Weight grams	Assay		Content		% of Pb. values
		Pb. %	Ag. ozs.	Pb. gms.	Ag. gm.-ozs.			Pb. %	Ag. ozs.	Pb. gms.	Ag. gm.-ozs.			Pb. %	Ag. ozs.	Pb. gms.	Ag. gm.-ozs.	
25	99	39·75	.....	39·4	.....	37·4	118	13·5	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
26	189	31·7	.....	.....	.....	55·0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
27	119	53·35	11·06	63·5	.....	58·9	132	8·96	.....	11·8	.....	10·0	712	4·70	.....	33·5	.....	31·1
28	96	50·7	.....	48·7	.....	44·8	162	10·0	.....	16·2	.....	14·9	.....	.....	.....	.....	.....	.....
29	131	46·5	.....	60·9	.....	56·2	202	8·5	.....	17·17	.....	15·8	.....	.....	.....	.....	.....	.....
30	134	49·6	10·36	66·5	1,393·6	61·2	86	13·2	.....	11·4	.....	10·3	780	3·97	.....	31·0	.....	28·5
31	148	51·8	.....	76·7	.....	70·5	173	6·4	.....	11·1	.....	10·2	679	3·10	.....	21·0	.....	19·3
32	142	53·5	.....	76·0	.....	70·0	129	7·0	.....	9·0	.....	8·3	761	3·20	.....	24·4	.....	21·6
33	125	59·8	.....	74·8	.....	68·9	86	7·9	.....	6·8	.....	6·3	816	3·3	.....	26·9	.....	24·8
34	91	56·8	10·4	51·7	946·4	47·6	110	13·3	8·96	14·6	.....	13·5	815	5·18	5·4	42·2	4,401	38·9
35	113	61·5	19·6	69·5	2,214·8	63·8	59	7·5	6·7	4·4	.....	4·0	856	4·1	4·0	35·1	3,430	32·2
36	134	50·3	16·4	67·4	2,197·6	61·7	168	8·9	7·4	14·9	.....	13·5	732	3·7	4·28	27·1	3,130	24·8

Test No. 145

A shipment of 351 pounds of lead-zinc sulphide ore was received on January 29, 1921, from the North Star mine, Kimberley, B.C.

The ore consisted of lead, zinc, and iron sulphides, and a siliceous gangue. The sulphide minerals represent about 90 per cent by weight, and the gangue about 10 per cent. The analysis of the ore was as follows:—

Lead (Pb)	18·00 %
Zinc (Zn)	30·45 "
Iron (Fe)	15·60 "
Silica ($Si O_2$)	7·50 "
Silver (Ag)	13·65 ozs.

The object of the experimental work was the separation of the sulphide minerals by preferential flotation, producing a marketable lead and zinc product.

A number of small scale tests were conducted, using oils, and the Callow Alphabetical flotation reagents. The results of the experimental tests, and the reagents used, are given in the following tables:—

REAGENTS USED

Test No. 1:

Flotation of lead—

2 pounds per ton Selecto added to ball mill.
5 pounds per ton lime added to ball mill.
Small quantity of steam distilled pine oil,
sufficient to produce a good froth, added to cells.

Flotation of zinc—

Lead tailings were dewatered.
5 pounds soda ash added to cells.
2 pounds No. 34 gravity fuel oil (Great Western Oil Co., Cleveland, O.).
Pine oil to froth.

Test No. 6:

Flotation of lead—

2 pounds per ton Selecto added to ball mill.
6 pounds per ton lime added to ball mill.
Small quantity of pine oil, sufficient to produce a good froth.

Flotation of zinc—

Lead tailings were dewatered.
6 pounds soda ash added to cells.
1 pound copper sulphate.
Pine oil to froth.

Test No. 7 :

Flotation of lead—

2 pounds per ton Selecto added to ball mill.
6 pounds per ton lime added to ball mill.
0·25 pound per ton Z-cake added to ball mill.
Small quantity of Aldol used as frother.

Flotation of zinc—

Lead tailings were dewatered.
6 pounds soda ash added to cells.
1 pound copper sulphate added to cells.
0·3 pound X-Y mixture.

Test No. 8:

Flotation of lead—

3 pounds per ton Selecto added to ball mill.
8 pounds per ton lime added to ball mill.
4 pounds per ton Na_2S (dry) added to flotation cells.
0·3 pound per ton pine oil.

Flotation of zinc—

Lead tailings were dewatered.
6 pounds per ton soda ash added to cells.
2 pounds per ton copper sulphate added to cells.
2 pounds per ton No. 34 gravity fuel oil.

Test No. 9:

Flotation of lead—

4 pounds per ton Selecto added to ball mill.
8 pounds per ton lime added to ball mill.
0·25 pound per ton Z-cake added to ball mill.
8 pounds per ton Na_2S added to ball mill.

Flotation of zinc—

Lead tailings were dewatered.
5 pounds per ton soda ash added to cells.
1 pound per ton copper sulphate added to cells.
0·3 pound per ton X-Y mixture added to cells.

Test No. 10:

Flotation of lead—

2 pounds per ton Selecto added to ball mill.
5 pounds per ton lime added to ball mill.
0·3 pound per ton No. 5 pine oil added to cells.

Flotation of zinc—

Lead tailings were dewatered.
5 pounds per ton soda ash added to cells.
2 pounds per ton copper sulphate added to cells.
1 pound per ton No. 34 gravity fuel oil.
0·1 pound per ton No. 5 pine oil.

Results of Tests

Test No.	Flotation products	Weight grams	Analysis			Content			Percentage of values in heads		
			Pb. %	Zn. %	Ag. ozs.	Pb. gms.	Zn. gms.	Ag. gm.-ozs.	Pb.	Zn.	Ag.
1	Lead concentrate	223	60·3	8·5	42·00	134·47	18·95	9,366	76·2	6·5	69·6
	Zinc concentrate	545	7·0	45·9	6·60	38·15	250·15	3,597	21·6	86·6	26·7
	Tailing	232	1·6	8·5	2·10	3·71	19·72	487	2·1	6·8	3·6
	Heads	1,000	17·63	28·88	13·45	176·33	288·82	13,450	99·9	99·9	99·9
6	Lead concentrate	342	38·0	22·5	.....	129·96	76·95	.....	74·3	25·7	.....
	Zinc concentrate	459	8·3	44·0	.....	38·10	201·96	.....	21·8	67·4	.....
	Tailing	208	3·2	10·0	.....	6·65	20·80	.....	3·8	6·9	.....
	Heads	1,009	17·47	29·97	.....	174·71	299·71	.....	99·9	100·0	.....
7	Lead concentrate	379	42·2	22·8	.....	159·94	86·41	.....	86·8	29·4	.....
	Zinc concentrate	430	4·8	44·7	.....	20·64	192·21	.....	11·2	65·4	.....
	Tailing	195	1·9	7·9	.....	3·70	15·40	.....	2·0	5·2	.....
	Heads	1,004	18·35	29·28	.....	184·28	294·02	.....	100·0	100·0	.....
8	Lead concentrate	245	53·73	13·52	.....	131·64	33·12	.....	72·9	11·2	.....
	Zinc concentrate	554	8·14	45·67	.....	45·10	253·04	.....	25·0	85·5	.....
	Tailing	205	1·83	4·73	.....	3·75	9·70	.....	2·1	3·3	.....
	Heads	1,004	17·98	29·47	.....	180·49	295·86	.....	100·0	100·0	.....
9	Lead concentrate	301	43·65	16·59	.....	131·39	49·94	.....	80·4	16·7	.....
	Zinc concentrate	452	5·76	48·51	.....	26·04	219·27	.....	15·9	73·4	.....
	Tailing	258	2·30	11·47	.....	5·93	29·59	.....	3·6	9·9	.....
	Heads	1,011	16·16	29·55	.....	163·36	298·80	.....	99·9	100·0	.....
10	Lead concentrate	255	59·83	14·15	.....	152·57	36·08	.....	81·2	11·9	.....
	Zinc concentrate	477	5·60	49·68	.....	26·71	236·97	.....	14·2	78·5	.....
	Tailing	275	3·16	10·49	.....	8·69	28·85	.....	4·6	9·6	.....
	Heads	1,007	18·66	29·98	.....	187·97	301·90	.....	100·0	100·0	.....

NOTE: Iron content in zinc concentrate test No. 1, was 12·3%.

SUMMARY AND CONCLUSIONS

From the foregoing table of results it has been determined that the sulphide mineral constituents of the ore can be separated by preferential flotation, and satisfactory lead and zinc products produced with a good recovery of the values in the ore. No difficulty was experienced in obtaining the separation, provided that fresh ore was used in conducting the tests. Fresh ore seems to be essential, as is shown by the examination of the results of tests Nos. 6, 7, 8, and 9. The lump ore received was crushed to ¼ inch, and sampled. A portion was cut out, ground to 20 mesh for test purposes, and used in tests Nos. 1 to 9 inclusive. Tests Nos. 1 to 5 inclusive were run immediately, and gave uniform results similar to those shown under test No. 1 in the table. After a month's time, further work was resumed on the portion ground to 20 mesh, for test purposes, and it was found that the results of the preceding tests could not be duplicated under the same conditions. Examples of this are given in the results of tests Nos. 6 and 7.

The conclusion was reached that the lead had become slightly oxidized. A little sodium sulphide was added, which improved the results, as shown in tests Nos. 8 and 9. In order to verify this conclusion and check results of tests Nos. 1 to 5, a fresh sample of ore was prepared from the ¼ inch material, and a test conducted under the same conditions. The results are given under test No. 10, and check closely with those of test No. 1.

The use of the Alphabetical reagents seems to give a better recovery of the lead, while the oil gives the higher recovery of the zinc. In the tests conducted with the Alphabetical reagents, the use of the reagent Selecto governs the separation of the lead and zinc. The lime prevents the iron, and to some extent, the zinc, from floating. After the flotation of the lead the pulp must be dewatered to eliminate the effect of the Selecto reagent on the subsequent flotation of the zinc. It was found that to obtain a high recovery of the zinc, it was necessary to use soda ash or caustic soda. The use of copper sulphate was found to be beneficial, but not essential. To obtain the best results, the lime should be added in sufficient quantities when the ore is being ground, before the flotation of the lead, otherwise it will be found that the iron sulphides will have a greater tendency to float with the zinc sulphides.

In the tests conducted, the ore was ground wet in a ball mill, to pass a 150 mesh screen. It is doubtful if good results could be obtained on coarser material.

(5)

COMPARATIVE TESTS ON COBALT SILVER SLIMES

C. S. Parsons and R. K. Carnochan

Test No. 146

In continuation of the experiments made in 1917 on the use of Canadian wood oils for the concentration of the values in the Cobalt slime tailing, a number of experimental tests were conducted on the use of the Rex, Alphabetical, and other reagents, on a shipment of 2,000 pounds of sand tailings, received at the Ore Dressing and Metallurgical Laboratories, February 9, 1921, from the Coniagas Mines, Limited, Cobalt, Ont. The results of these tests are given in the following tables:—

TABLE I

Results of Tests using Rex Flotation Reagents

Test No.	Heads ozs. Ag.	Conc. ozs. Ag.	Midds. ozs. Ag.	Tails ozs. Ag.	Conc. ratio	Recovery
38	8·08	66·60	.....	5·88	1 : 26·3	30·8
39	8·08	88·37	.....	6·38	1 : 44·3	24·3
41	8·08	74·72	.....	4·90	1 : 23·8	40·2
42	8·08	82·12	.....	4·44	1 : 22·7	46·1
43	8·08	72·44	.....	3·90	1 : 17·0	53·8
73	3·70	328·20	20·46	2·24	1 : 144·2	53·9
74	3·70	251·75	18·00	2·42	1 : 104·4	52·8

The above tests were run under the following conditions, except where differences are noted.

Janney flotation machine and Rex flotation reagents used. Five hundred grams of silver slimes ground to 100 mesh, mixed in pebble mill 5 minutes, agitated in machine 5 minutes, and floated for 8 minutes.

Test No.
- 41 Ruth machine, 1,000 grams.
- 42 Ruth machine, 1,000 grams.
- 43 Ruth machine, 1,000 grams.
- 73 Four charges run, and concentrates combined and re-run to clean them up. In re-running, mixed 5 minutes, agitated 5 minutes, and floated for 5 minutes.
- 74 Same as 73, except that concentrates were floated 10 minutes when re-cleaning.

TABLE II

Results of Tests using Alphabetical and Rex Flotation Reagents

Head sample of shipment assayed 3·48 ounces of silver per ton.

If not otherwise stated, the ore in each of the following tests was crushed dry to pass 60 mesh, and then 1,000 grams charged to a small ball mill, ground with one part of water to two parts of ore, for 30 minutes.

Test No.	Re-agents used, etc.	Concentrate		Middling		Tailing	
		Wt. gram	Assay oz. Ag.	Wt. gram	Assay oz. Ag.	Wt. gram	Assay oz. Ag.
1	1 pound per ton X. Y.....	19	114·3	96	5·02	893	0·80
2	2 " soda ash 1 " X. Y.....	46	52·70	128	1·72	823	0·98
3	4 " soda ash 1 " X. Y.....	20	122·4	61	4·50	913	0·88
4	6 " soda ash 1 " X. Y.....	18	78·51	91	3·10	877	1·24
5	6 " lime 1 " X. Y.....	32	60·64	113	2·40	858	1·70
6	2 " lime 1 " X. Y.....	54	42·30	175	1·30	771	1·28
7	2 " soda ash 2 " Rex No. 12.....	40	19·08	182	4·14	756	2·76
8	3 " Rex No. 12.....	26	23·22	134	4·14	840	3·04
9	4 " soda ash 2 " Rex No. 12.....	36	17·76	158	4·14	794	2·86
10	4 " sodium sulphide 2 " Rex No. 12.....	50	8·32	.....	.....	923	3·30
11	4 " soda ash 2 " Rex No. 12 1 c. c. ammonia.....	27	28·0	118	3·22	839	2·58
12	4 pound per ton soda ash 2 " Rex No. 12 1 c. c. ammonia.....	45	13·40	126	4·04	790	3·34
13	0·2 pounds per ton pine oil..... 1·6 " C.T. creosote..... 0·2 " coal oil.....	70	33·04	143	3·34	764	0·70
14	4 pounds per ton soda ash 1 " NH_4OH 2 " Rex No. 12..... ground sands for 45 minutes.....	39	17·08	190	3·38	751	2·78
15	10 pounds per ton soda ash 2 " Rex No. 12 1 pound per ton soda silicate for reclean-ing concentrate.....	49	15·58	163	3·64	759	2·96
16	3 pounds per ton sod. hydroxide 2 " Rex No. 12.....	68	13·18	274	3·02	633	2·82

(6)

THE MILLING OF ASBESTOS ROCK

R. K. Carnochan

The general milling practice for the recovery of the asbestos fibre from the serpentine rock has not changed to any extent since its innovation. It is true that marked improvements have been made, resulting in more efficient recovery of the fibre, but production of the fibre in marketable grades has claimed the attention of the operators, and little attention has been given to experimental work on new processes for the recovery of the fibre from the rock. While metallurgical practice has radically changed in connection with the treatment of the metallic ores, no changes of such a nature have taken place in the milling of asbestos rock.

The general practice for the recovery of the fibre is a dry process. After as much crude fibre as is economically possible is picked out from the broken rock in the pits and the dry rock discarded, the remainder, the milling rock, is delivered to the mill for the separation and recovery of the fibre. The rock is broken in primary breakers, such as jaw and gyratory crushers, to about three inch size. It is then dried in rotary or vertical dryers. From this stage the process is one of breaking the rock to release the fibre, and at the same time fiberizing or fluffing the fibre, and the removal of the fibre by suction as the crushed material passes over shaking tables. This process is repeated until the rock is reduced to such a size that it contains only very short fibre, the recovery of which is not economically feasible. The breaking and fiberizing machines used are hammer crushers, rolls, Simons disc crushers, cyclone and jumbo fiberizers. The hammer crushers, cyclone and jumbo fiberizers were developed in the district as being adapted to the breaking and fiberizing of asbestos rock. The fibre is collected in fibre collectors, passed through dusters to remove dust and fine particles of rock adhering to it. It is then passed through graders, which grade it into the various commercial products. A standard asbestos testing machine was developed in the district, which is used by all operators, on which the grades are tested for length of fibre. Shipments are made conforming to certain tests on this machine.

The investigations carried on in the Ore Dressing and Metallurgical Laboratories were confined to the use of the conical ball mill for the cracking of the rock and liberation of the fibre after primary crushing. The use of the cyclone and jumbo fiberizers in the present practice is found to be costly as to power consumption and to maintenance and repairs. The violent action in these machines is also more or less destructive of the long fibre. Shipments were received of both the "slip fibre" variety of rock from East Broughton, Que., and the "cross fibre" variety from Black Lake, Que. The object of the experimental work was to determine if some other types of crushing machinery could not be used for the cracking of the rock and liberation of the fibre, with less destruction of the fibre than by the use of cyclone and jumbo fiberizers. In conducting this work many points of interest were brought out, which are noted under the description of the investigation.

It is proposed to carry the investigation further along the lines of wet crushing of the rock. As one of the chief items of cost, especially in certain seasons of the year, is that of drying the rock for the present practice, a considerable saving could be made if a wet process could be applied, with an improvement of the working conditions in the mills of the district.

Test No. 147

A shipment of 500 pounds of tailings was received on February 9, 1921, at the Ore Dressing and Metallurgical Laboratories, from the Asbestos Mines, Limited, East Broughton, Que.

This material represented the discard of their milling operations, and contained a considerable quantity of fine fibre not freed from the rock.

The purpose of the test work on this shipment was to determine if any products suitable for trade purposes could be obtained from the rock, what further treatment was necessary, and what use could be made of such products.

No experimental work was done on this shipment, as other work was more pressing, and it was found that the mills of the district were making a large quantity of fine fibre, for which there was not a brisk demand. It was decided advisable to await a more opportune time for conducting the work.

Test No. 149

Two shipments of asbestos rock were received at the Ore Dressing and Metallurgical Laboratories, from the Asbestos Mines, Ltd., East Broughton, Quebec. The rock was from the Boston mine, was received crushed to about three-inch size, the product of their secondary crushers, and represented the feed to the Jumbo and Cyclone crushers or fiberizers.

The rock from the Boston mine is of the "slip fibre" type, representative of the East Broughton deposits, which contains no crude, but is a straight milling rock. In this respect it is dissimilar to the Thetford mines, and Black Lake deposits. The rock is also much softer than that from the other districts.

The first shipment of three tons was received on March 30, 1921, and the second shipment of ten tons on May 16, 1921.

In the company's milling operations, the Jumbo and Cyclone crushers or fiberizers are used to separate the fibre from the rock. These machines, while being good fiberizers, are costly to operate, as to power consumed and repairs required. Moreover, the action is so violent that the fibre is more or less broken up, resulting in an excessive amount of fines. The object of the experimental work was to determine what results could be obtained from the Hardinge mill in comparison with the Jumbo and Cyclone crushers or fiberizers, as to amount and grade of fibre produced, and also as to power consumption and repairs. As no data were available or could be secured as to the efficiency of the crushers and fiberizers in use, the only course to pursue was to perform the tests and leave the operators to draw their own conclusions and comparisons of results.

The first shipment of three tons was used up in making adjustments to the Hardinge mill, to suit the crushing of this type of rock. Tests were run using pebbles and balls as the grinding or breaking media, increasing the speed of the mill, increasing the inclination of the mill, and the removal of the fibre from the mill, on being freed from the rock, by suction.

In the crushing of asbestos rock, the main point to consider is the separation of the fibre from the rock to obtain the maximum amount of long fibre. The objective is, therefore, to crush or break the rock in such a manner as to remove the fibre without grinding, and with as little damage to it as possible, and to remove it as soon as it is free, so that it will not be damaged by further crushing or breaking action. Up to the present, on account of its universal use as a grinding mill, it was considered that the Hardinge mill would not be adaptable to this class of work, in fact it was claimed that asbestos rock, due to its nature, would grind quite freely for a time and then pack on the lining until crushing action ceased almost entirely. Our tests on this first shipment proved this not to be the case. By elevating the feed end of the mill to obtain quick discharge; by using large balls to obtain coarse grinding; by increasing the speed of the mill to obtain a breaking and not a grinding action; and by connecting the discharge end of the mill with a suction to remove the fibre as it was freed, favourable results could be obtained. It was also determined that the rate of feed to the 4′ 6″ × 13″ mill should be about three tons per hour.

Having obtained these data, a new fibre collector was built, with connections to the discharge end of the mill and to a shaking screen, and also to an exhauster which exhausted into another collector and to the air. With this arrangement there was practically no loss of fibre. The ball mill was adjusted to what had been proven from the runs on the first shipment as the most desirable setting. It was also decided not to feed three-inch material to the mill, but to crush the rock in a jaw crusher and rolls to one-inch size before feeding to the ball mill, as this was the practice that was contemplated in the revised layout of the Boston mill. By crushing to one-inch size before feeding to the ball mill a large percentage of the fibre would be freed, and removed by suction.

Run No. 1.—Nine tons of the rock from shipment No. 2 were weighed, crushed in a jaw crusher to 1½″, and reduced to 1″ in rolls. This operation gave the following:—

Product	Weight pounds	Per cent of heads
– 1″	17,874·5	99·30
Dust loss	125·5	0·70
Total	18,000·0	100·00

Run No. 2.—The rock reduced to 1″ was sampled by an automatic Vezin sampler, the main flow passing over a shaking screen, fitted with a blank screen for the first half and with a $\frac{1}{16}$″ slot screen for the second half at the discharge end. Two suction pipes into a main header to the collector were placed over the screen for the removal of the fibre. An

exhauster was placed between the first collector and discharged into a second collector from which a second exhauster discharged into the air. Any fibre not collected in the first collector was deposited in the second one. With this arrangement there was practically no loss of fibre. This operation gave the following:—

Product	Weight Pounds	Per cent of heads	Test
+1/16"	12,150·0	68·24	
−1/16"	3,312·0	18·60	
Fibre—1st collector	2,070·0	11·65	0—0·5 —4·7 —10·8
Fibre—2nd collector	21·0	0·12	(Large per cent rock)
Dust loss	123·5	0·69	
Sample	194·0		
Feed	17,874·5	99·30	

Run No. 3—First ball mill run.—The + $\frac{1}{16}$" material from Run No. 2 was sampled by an automatic Vezin sampler and fed to a Hardinge mill at the rate of 2·75 tons per hour. The 4′ 6″ x 13″ mill was run at 36 r.p.m. and carried a load of 2,900 pounds of 4″ steel balls. The feed end of the mill was raised three inches from the horizontal. The discharge end was fitted with a suction pipe to the first collector, with a T pipe close to the discharge, permitting of the regular discharge and also of the fibre being sucked over into the collectors. This first ball mill run gave the following:—

Product	Weight pounds	Per cent of heads
Fibre—1st collector	6,163	35·04
Dust—2nd collector	113	0·64
Rock, discharged	5,540	31·49
Rock, in mill	188	1·07
Sample	146	
Feed	12,150	68·24

Run No. 4—Second ball mill run.—The rock discharged from the first run of the ball mill was sampled by an automatic Vezin sampler and fed back to the mill, which contained some rock from the preceding operation. The mill was run under the same conditions. After the completion of the run the mill was dumped. This operation gave the following:—

Product	Weight pounds	Per cent of heads
Fibre—1st collector	1,377·0	8·04
Dust—2nd collector	38·5	0·22
Tailing, discharged	3,476·0	20·30
Tailing, from mill	553·0	3·23
Dust loss	131·0	0·77
Sample	137·5	
Feed	5,713·0	32·56

Run No. 5—Fibre from first ball mill run.—After sampling the fibre from Run No. 3, or the first ball mill run, it was run over the shaking screen to remove the rock and dust. This gave the following products:—

Product	Weight pounds	Per cent of heads	Test
Fibre—1st collector	940	5·41	0—0 —3·2 —12·8 (fairly clean)
Dust—2nd collector	14	0·08	
+1/16" rock	2,678	15·40	
−1/16"	2,360	13·57	
Dust loss	101	0·58	
Sample	70		
Feed	6,163	35·04	

The $+\frac{1}{16}''$ rock contained considerable unfiberized material and should have been returned to the circuit.

The $-\frac{1}{16}''$ material contains some fibre that would produce a marketable grade as shown in a following table, under Run No. 12.

Runs Nos. 6, 7 and 8—Fibre from second ball mill run.—The fibre from Run No. 4, or the second ball mill run, was used in these three runs, an attempt being made to remove the rock and bring the fibre up to a suitable grade. In Run No. 6, the fibre from Run No. 4 was run over the shaking screen, using a 12×12 mesh screen, and with the suction at the end of the screen near the discharge, so as to allow the fines to screen out from the fibre before it was sucked up into the collectors. In Run No. 7, the fibre from Run No. 6 was re-run, and in Run No. 8 the fibre from Run No. 7 was re-run. The second collector was not cleaned out until the finish of Run No. 8. The following are the products obtained:—

Product	Weight pounds	Per cent of heads	Test
Fibre—1st collector	180·0	1·05	0 —0 —2·8 —13·2
Dust—2nd collector	100·0	0·58	
+12, run 6	443·5	2·60	
+12, run 7	25·0	0·15	
+12, run 8	29·0	0·17	
−12, run 6	230·0	1·35	
−12, run 7	109·0	0·64	
−12, run 8	194·0	1·14	
Dust loss	61·5	0·36	
Feed	1,372·0	8·04	

The +12 product from Run No. 8 contained considerable unfiberized material. A test was made on this product, the results of which are shown in the following table of fibre recovered:—

Run No. 9—On fibre from initial crushing in jaw crusher and rolls to one inch.—The fibre from Run No. 2 was run over the shaking screen fitted with a 12×12 mesh screen, as in Run No. 8. The object of this run was to improve the grade. The following products were made:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector	900	5·09	0—0·5—4·5—11·0
Dust, 2nd collector	5	0·03	(fairly clean).
+12	376	2·12	
−12	746	4·22	
Dust loss	34	0·19	
Feed	2,061	11·65	0—0·5—4·7—10·8 (large per cent rock).

Run No. 10—On $-\frac{1}{16}''$ material from initial crushing.—The $-\frac{1}{16}''$ material from Run No. 2 was re-run over the shaking screen fitted with a 12×12 mesh screen to recover any fibre which passed through this screen. This run would not have been necessary if the square mesh screen had been used instead of the $\frac{1}{16}''$ slotted screen. The results were as follows:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector	204	1·15	0—0—1·9—14·1
Dust, 2nd collector	2	0·02	(clean).
+12	1,098	6·17	
−12	1,957	10·98	
Dust loss	49	0·28	
Feed	3,312	18·60	

Run No. 11—Re-run of fibre from first ball mill run.—The fibre from Run No. 5 was re-run over the shaking screen to remove any further rock and dust from it. The 12×12 mesh screen was used. The products from this run were as follows:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector	475	2·75	0—0—4·1—11·9
Dust, 2nd collector	6	0·03	(clean).
+12	21	0·12	
−12	417	2·41	
Dust loss	17	0·10	
Feed	936	5·41	0—0—3·2—12·8 (fairly clean).

Run No. 12—On $-\frac{1}{16}$" material from first ball mill run.—The $-\frac{1}{16}$" material from Run No. 5, was run over the shaking screen fitted with a 12 × 12 mesh screen to recover any coarse fibre which passed through the $-\frac{1}{16}$" slotted screen. Had the square mesh screen been used in Run No. 5, this run would not have been necessary. The following products were obtained:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector	134	0·77	0·—0—1·5—14·5
Dust, 2nd collector	4	0·02	(clean).
+12	492	2·81	
−12	1,737	9·96	
Dust loss	1	0·01	
Feed	2,368	13·57	

Runs Nos. 13 and 14—Re-run of fibre from initial crushing.—These two runs were made on the fibre from Run No. 9, to remove any dust and rock remaining in it. Both runs were made over the shaking screen, using a 12 × 12 mesh screen. Run No. 13 did not remove sufficient of the rock, so the fibre was re-run over the screen a second time. The products were as follows:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector run No. 14	366	2·07	0—1·8—5·6—8·6
Dust, 2nd collector runs Nos. 13 and 14	11	0·06	(clean).
+12, run No. 13	114	0·64	
+12, run No. 14	21	0·12	
−12, run No. 13	257	1·45	
−12, run No. 14	114	0·65	
Dust loss, runs Nos. 13 and 14	17	0·10	
Feed	900	5·09	0—0·5—4·5—11·0 (fairly clean).
Fibre, 1st coll. run No. 13	509	.....	0—0·9—4·9—10·2 (fairly clean).

The +12 material from Runs Nos. 13 and 14 contained a considerable amount of unfiberized material. These products should have been returned to milling operations. Test runs were made on these, the results of which are given in the table of fibre recovered.

Runs Nos. 15 and 16—On fibre from Run No. 14.—These two short runs were made on a portion of the fibre from Run No. 14, to determine if it would be possible to bring some of the fibre up to a high grade, and also to determine if it was possible to figure out fairly accurately what grades of fibre could be produced from a given lot of fibre.

The two runs were made by passing the fibre over a ¼" shaking screen, the screen being used as a grader. Run No. 16 is on the +¼" fibre from Run No. 15.

Products Run No. 15	Weight pounds	Per cent of heads	Test
+¼-in. fibre	6·00	0·36	0·5+5·0+5·2+5·3 (clean).
−¼-in. fibre, 2nd hopper	13·25	0·79	0·0+1·0+7·1+7·9 (clean).
−¼-in. fibre, 1st hopper	15·50	0·92	0·0+0·3+6·9+9·7 (clean).
Feed, run No. 15	34·75	2·07	0·0+1·8+5·6+8·6 (clean).

Products Run No. 16	Weight pounds	Per cent of heads	Test
	lbs.		
+¼-in. fibre	1·60	0·10	2·2+7·6+2·8+3·4 (clean).
−¼-in. fibre	4·40	0·26	0·0+3·8+5·6+6·6 (clean).
Feed, run No. 16	6·00	0·36	0·5+5·0+5·2+5·3 (clean).

It is here demonstrated that it is possible to produce a high grade fibre by grading. If in each of these runs the figures given in the fibre test on the products are multiplied by the weight of the product, and corresponding terms added, and the sum divided by the weight of the feed, it will be found to give very nearly the fibre test of the feed. Hence, we may conclude that it is possible to figure fairly accurately what grades can be produced from a certain fibre, and also that it is correct in figuring recovery, to sum up from the different fibre products the percentage of heads on the same testing screen.

The following table shows the different fibres recovered from all the different runs, etc. In the list will be noticed some +12 products. These are really middlings which would be returned to the circuit in regular mill work. Small tests have been conducted on these products, and whatever fibre was found possible to recover from them has been tabulated.

Product	Per cent of heads	Per cent fibre	Per cent re-covered	Testing				Per cent heads recov'd.			
				+2	+4	+10	−10	+2	+4	+10	−10
Fibre run 8	.....	.....	1·05	.....	.....	2·8	13·2	.....	.....	0·184	0·866*
" 10	.....	.....	1·15	.....	.....	1·9	14·1	.....	.....	0·137	1·013
" 11	.....	.....	2·75	.....	.....	4·1	11·9	.....	.....	0·705	2·045*
" 12	.....	.....	0·77	.....	.....	1·5	14·5	.....	.....	0·072	0·698*
" 14	.....	.....	2·07	.....	1·8	5·6	8·6	.....	0·233	0·725	1·112
+12 run 8	0·17	17·5	0·03	.....	0·1	2·2	13·7	.....	.....	0·004	0·026*
" 9	2·12	12·9	0·27	.....	0·3	3·6	12·1	.....	0·005	0·061	0·204
" 11	0·12	37·9	0·05	.....	0·2	5·8	10·0	.....	0·001	0·018	0·031*
" 13	0·64	60·0	0·38	.....	0·2	5·8	10·0	.....	0·005	0·138	0·237
" 14	0·12	80·0	0·10	.....	0·2	5·8	10·0	.....	0·001	0·036	0·063
Total fibre	.....	.....	8·62	.....	0·5	3·8	11·7	.....	0·245	2·080	6·295

NOTE.—Products marked * are from ball milling.

A small sample of the feed to the first ball mill run, Run No. 3, was separated by screening into a number of different sizes. As much fibre as possible was picked out of the larger sizes, and sucked out of the smaller ones. The sizes were all mixed and ground in a small ball mill, and the product screened, picked, and sucked, as before. This milling, screening, picking, and sucking was continued until the rock was clean of fibre. The fibre obtained was tested in the usual manner, and the following table shows the results of this small test compared with what was obtained from the large runs:—

Product	Per cent of heads	Per cent fibre	Per cent re-covered	Testing				Per cent heads recover'd			
				+2	+4	+10	−10	+2	+4	+10	−10
Feed run 3	68·24	12·87	8·78	.....	0·1	0·9	15·0	.....	0·055	0·494	8·231
Fibre from mill... (large run).	.....	.....	4·65	.....	.....	3·4	12·6	.....	0·983	0·001	3·666

In this table of comparison, it is noticed that the small test recovered much more +4 than the large runs. This was to be expected, as in hand picking in the small test, a knife was used to cut off the fibre from the rock, and much unfiberized asbestos was put with the picked material.

It will also be noticed that the large runs give the largest amount of total fibre, +10. This would indicate that the Hardinge ball mill, with suction, is a very effective machine for the work upon which it was employed in the larger runs.

We are unable to make a comparison of the efficiency of the Hardinge ball mill with the Cyclone or Jumbo fiberizers, as no data are available on the work of the latter machines.

From the results given above, however, the mill operator may be able to draw some conclusions on the relative efficiency of the two types of mills.

Test No. 152

A shipment of ten tons of asbestos rock was received at the Ore Dressing and Metallurgical Laboratories, May 27, 1921, from the Black Lake Asbestos and Chrome Co., Ltd., Black Lake, Quebec.

The rock was of the cross fibre variety, representative of the Black Lake type of rock, and much harder than the slip fibre rock of the East Broughton deposits. As received it was composed of about three-inch material, with the fines removed, and as the fines from the primary crushing carry much more fibre than the coarse when reduced to this size, the percentage of fibre was, therefore, lower than the average mill feed, and the rock would also contain shorter fibre, the longer being freed by the primary crushing, and entering the fines.

The rock received represented the coarser material fed to the Cyclone and Jumbo fiberizers. It was desired that test work be conducted on this rock for the recovery of the fibre, and to obtain some data with respect to the crushing and liberation of the fibre in a Hardinge mill, as compared with the operations of the fiberizers in use for this purpose.

It is a fairly well established fact that the rock should be reduced finer than three inch before feeding to the fiberizers, and that this reduction can be accomplished in such crushing devices as jaw crushers, gyratories, rolls, hammer crushers, etc., without great injury to the fibre. The three-inch material received was, therefore, reduced to 1½″ and 1″ size before feeding to the Hardinge mill.

As no data were available as to the efficiency of the Cyclone and Jumbo fiberizers, no comparative results could be noted. The following tests, however, give the results obtained by the use of the Hardinge mill on material finer than 1½″.

Test No. 1

Run No. 1.—Five tons of the asbestos rock was weighed, crushed in a jaw crusher to 1½″, and sampled by an automatic Vezin sampler, the main flow passing over a Ferraris shaking screen fitted with a blank screen for the first half, and with a 12 mesh square screen for the second half, at the discharge end. One suction pipe into a main header to a collector was placed over the screen at the discharge end of the screen for the removal of the fibre. An exhauster was placed between the first collector and discharged into a second collector, from which a second exhauster discharged into the air. Any fibre not collected in the first collector was deposited in the second one. With this arrangement there was practically no loss of fibre. Any fibre deposited in the second collector was extremely

fine, and of no value. The fibre deposited in the first collector was re-run over the shaking screen twice, to remove fines and pieces of rock, and unfiberized asbestos. These operations gave the following results:—

Product	Weight pounds	Per cent of heads	Test
Fibre	18	0·182	1·1—5·6—6·5—2·8
—12 (fines from screen)	666	6·729	
+12 (tailings from screen)	9,114	92·089	
Dust (2nd collector and loss)	99	1·000	
Feed	9,897	100·000	

Run No. 2—First ball mill run.—The +12 mesh material from run No. 1 was sampled by an automatic Vezin sampler and fed to a Hardinge mill at the rate of 2·5 tons per hour. The 4′ 6″ × 13″ mill was run at 35 r.p.m. and carried a load of 3,000 pounds of 4″ steel balls. The feed end of the mill was raised three inches from the horizontal. The discharge end was fitted with a suction pipe to the first collector, with a T pipe close to the discharge, permitting of the regular discharge, and also of the fibre being sucked over into the collectors. The fibre from the first collector was run over the shaking screen twice to remove the rock and dust. The following are the products obtained:—

Product	Weight pounds	Per cent of heads	Test
Fibre	29·0	0·296	0·0—2·4—4·4—9·2
—12 (from screen)	704·5	7·195	
+12 (tailings from mill and screen)	8,261·0	84·364	
Dust (2nd collector and loss)	23·0	0·234	
Feed	9,017·5	92·089	

Run No. 3—Second ball mill run.—The +12 mesh material from run No. 2 was sampled by an automatic Vezin sampler and fed back to the mill, which contained some rock from the preceding operation. The mill was run under the same conditions. The fibre from the first collector was run over the shaking screen twice to remove the rock and dust. In this mill run, the mill tailings were found to contain quite a quantity of fibre which had not been removed by the suction, so they were run over the shaking screen to get this fibre out, and the fibre itself was cleaned by passing it over the shaking screen once. At this stage of the test work it was observed that the bags that had contained the asbestos rock were covered inside with quite a quantity of long fibre. The bags were all turned inside out and the fibre removed. This fibre from the bags was

cleaned up by passing it over the shaking screen twice. The result of all these operations follows:—

Product	Weight pounds	Per cent of heads	Test
Fibre (from mill suction)	20	0·206	0·0—2·9—3·2—9·9
Fibre (from mill tails)	24	0·248	1·4—5·5—6·2—2·9
Fibre (from bags)	4	0·041	5·1—3·3—3·5—4·1
−12 (from screen)	3,217	33·199	
+12 (from screen)	4,889	50·453	
Dust (2nd collector and loss)	21	0·217	
Feed	8,175	84·364	

Run. No. 4—Third ball mill run.—The +12 material from run No. 3 was sampled by an automatic Vezin sampler and fed back to the mill, which contained some rock from the preceding operation. The mill was run under the same conditions. The fibre from the first collector was run over the shaking screen twice to remove the rock and dust. The tailings from the mill contained some fibre, and this fibre was removed by passing the tailings and the cleanout of the mill over the shaking screen. The fibre obtained in this way was cleaned up by passing it over the shaking screen twice. On the completion of this run the second collector was cleaned out along with the pipes leading to it, and 61·5 pounds of dust obtained. The dust produced in the work up to this point was 339 pounds. The third ball mill run gave the following:—

Product	Weight pounds	Per cent of heads	Test
Fibre (from mill suction)	4·5	0·047	0·0—2·6—4·1—9·3
Fibre (from mill tails and clean out)	6·5	0·069	0·3—4·3—7·4—4·0
−12 (from screen)	2,113·0	22·354	
+12 (from screen)	2,449·0	25·909	
Dust (2nd collector and loss)	196·0	2·074	
Feed	4,769·0	50·453	

The following table shows the amount of fibre obtained in all the runs of Test No. 1:—

FIBRE OBTAINED—TEST No. 1

Run No.	Per cent of heads	Test	Per cent of heads			
			+2	+4	+10	−10
Run No. 1	·182	1·1—5·6—6·5—2·8	·012	·064	·074	·032
Run No. 2	·296	0·0—2·4—4·4—9·2	·0	·044	·082	·170
Run No. 3 (mill suction)	·206	0·0—2·9—3·2—9·9	·0	·307	·041	·128
Run No. 3 (mill tails)	·248	1·4—5·5—6·2—2·9	·022	·085	·096	·045
Run No. 3 (from bags)	·041	5·1—3·3—3·5—4·1	·013	·008	·009	·011
Run No. 4 (mill suction)	·047	0·1—2·6—4·1—9·3	·0	·008	·012	·027
Run No. 4 (mill tails)	·069	0·3—4·3—7·4—4·0	·001	·019	·032	·017
Total fibre	1·C89	0·7—3·9—5·1—6·3	·C48	·265	·346	·430

Test No. 2

Run No. 1.—The balance of the shipment of asbestos rock (10,149 pounds) was crushed in a breaker to pass ¾", and sampled by an automatic Vezin sampler, the main flow passing to the shaking screen, where the fibre was sucked out of it. This fibre was then cleaned by passing it over the shaking screen once. The following are the products:—

Product	Weight pounds	Per cent of heads	Test
Fibre	28·5	0·284	0·5—4·6—6·4—4·5
−12 (from screen)	1,213·0	12·104	
+12 (tailings from screen)	8,552·0	85·342	
Dust (2nd collector and loss)	227·5	2·270	
Feed	10,021	100·000	

Run No. 2—Ball mill run.—The +12 material from run No. 1 was split into two portions by shovelling. One of these portions was approximately four-fifths of the whole, and this was used for run No. 2. The larger portion was fed to the ball mill under the same conditions as the previous ball mill runs, except that no suction was used on the mill. The discharge from the mill was run over the shaking screen to remove the freed fibre. After the whole portion had been fed to the mill and the discharge from the mill put over the screen, the +12 from the screen was fed back to the mill and the mill tailings treated as before. This series of operations was repeated five times, when it was found that the +12 from the screen contained no appreciable amount of fibre. The fibre from the first collector was then cleaned up by passing it over the shaking screen once. This work gave the following results:—

Product	Weight pounds	Per cent of heads	Test
Fibre	204·5	2·558	
−12 (from screen)	2,316·0	28·971	
+12 (tailings from screen)	4,147·5	51·880	
Dust (2nd collector and loss)	154·5	1·933	
Feed	6,822·5	85·342	

The following is a summary of the fibre obtained in Test No. 2:—

FIBRE OBTAINED—TEST No. 2

Run No.	Per cent of heads	Test	Per cent of heads			
			+2	+4	+10	−10
Run No. 1	·284	0·5 4·6 6·4 4·5	·009	·082	·113	·080
Run No. 2	2·558	0·0 0·3 8·4 7·3	·0	·048	1·343	1·167
Total fibre	2·842	0·1 0·7 8·2 7·0	·009	·130	1·456	1·247

Test No. 3

This test was made on approximately one-fifth of the +12 from test No. 2, run No. 1. In calculating this test, it is necessary to consider that a run has already been made on the material used, so we must call the first actual run, run No. 2, and for run No. 1, take the results obtained in run No. 1, test No. 2.

Run No. 2.—The +12 material was crushed, sampled by an automatic Vezin sampler, and fed to the shaking Ferraris screen, the first half of the table being fitted with a 12 mesh square screen, and the second half with a $\frac{1}{8}''$ slotted screen. The suction pipe to remove the fibre was placed over the end of the 12 mesh screen where it joined the $\frac{1}{8}''$ screen. The $+\frac{1}{8}''$ product was crushed and fed back to the screen until everything passed $\frac{1}{8}''$. The fibre produced was cleaned by passing it over the screen once. These operations gave:—

Product	Weight pounds	Per cent of heads	Test
Fibre	9	0·469	0·7—5·5—4·7—5·1
$-\frac{1}{8}''+12$ (from screen)	878	45·745	
−12 (from screen)	730	38·034	
Dust (2nd collector and loss)	21	1·094	
Feed	1,638	85·342	

Run No. 3.—This run was made on approximately one-eighth of the +12 material from run No. 2. The +12 was crushed and fed to the shaking screen fitted up as in run No. 2, the +12 being re-crushed and fed back until all passed through the 12 mesh screen. The fibre obtained was cleaned by passing over the screen once. The following are the products:—

Product	Weight pounds	Per cent of heads	Test
Fibre	0·25	0·102	0·0—0·4—4·0—11·6
−12 (from screen)	112·00	45·542	
Dust (2nd collector and loss)	0·25	0·101	
Feed	112·50	45·745	

The following is a summary of the fibre obtained from the different runs of test No. 3:—

FIBRE OBTAINED—TEST No. 3

Run No.	Per cent of heads	Test	Per cent of heads			
			+2	+4	+10	−10
Run No. 1	·284	0·5—4·6—6·4— 4·5	·009	·082	·113	·080
Run No. 2	·469	0·7—5·5—4·7— 5·1	·021	·161	·138	·149
Run No. 3	·102	0·0—0·4—4·0—11·6	.....	·003	·025	·074
Total fibre	·855	0·6—4·6—5·1— 5·7	·030	·246	·276	·303

Test No. 4

This is a test on 51 pounds of the head sample of test No. 1, which was made to see how the heads of test No. 1 compared with the heads of test Nos. 2 and 3. The test was made in the same manner as test No. 3, the rock being crushed to $\frac{3}{4}''$, then to $\frac{1}{8}''$, and then to 12 mesh, the fibre being removed during the crushing, and cleaning up after each stage of crushing had been completed.

Product	Weight pounds	Per cent of heads	Test
Fibre from crushing to $\frac{3}{4}''$	0·114	0·224	2·5—4·2—5·1— 4·2
" " $\frac{1}{8}''$	0·178	0·349	3·4—5·3—3·2— 4·1
" " 12 mesh	0·024	0·047	0·0—0·0—5·8—10·2
—12 from crushing to $\frac{3}{4}''$	5·500	10·784	
" " $\frac{1}{8}''$	23·500	46·079	
" " 12 mesh	19·750	38·725	
Dust (2nd collector and loss)	1·934	3·792	
Feed	51·000	100·000	

FIBRE OBTAINED—TEST No. 4

Product	Per cent of heads	Test	Per cent of heads			
			+2	+4	+10	−10
Fibre, crushing to $\frac{3}{4}''$	·224	2·5—4·2—5·1— 4·2	·035	·059	·071	·059
" " $\frac{1}{8}''$	·349	3·4—5·3—3·2— 4·1	·074	·116	·070	·089
" " 12 mesh	·047	0·0—0·0—5·8—10·2	.....	.....	·018	·030
Total fibre	·620	.....	·109	·175	·158	·178

After finishing test No. 4, the second collector and pipes leading to it were cleaned out and 209 pounds of dust obtained. This dust was from tests Nos. 2, 3 and 4. In these tests the total dust produced was 405 pounds.

CONCLUSIONS

An examination of the fibre produced in tests Nos. 3 and 4, shows that although the amounts of the different sizes recovered show a great difference, still the +2 and +4 added together in each test give practically the same figures, and the +2, +4 and +10 added in each test, are not a great deal apart.

Making a suitable allowance for test No. 4, being on such a small amount of rock, one can conclude that the feeds to tests Nos. 1, 2 and 3 are the same.

Test No. 3 was carried on in such a way as to give practically all the long fibre that it would be possible to extract from the rock, and the results of this test can be used as a standard to compare with tests Nos. 1 and 2.

On comparing test No. 1 with test No. 3, we see that test No. 1 recovered more fibre in all the sizes, and from this can conclude that the Hardinge mill operated with suction is a very suitable machine to treat the kind of asbestos rock submitted.

Comparing the results of test No. 2 with those of test No. 3, we find that the +2 and +4 recovered are both less in test No. 2 than in test No. 3, and that the +10 and −10 are both more. From this we can conclude that the Hardinge mill with suction is superior to the same mill without suction, and that without suction the result would seem to be to grind up the long fibre into shorter material.

The results of the test work show that the Hardinge mill operated under favourable conditions can be used for the crushing of asbestos rock of the "cross fibre" variety. It is not possible to give a comparison of efficiency over the fiberizers in use, as no data are available as to results from these particular machines.

The results also show that in order to obtain the greatest percentage of long fibre, stage crushing is necessary, with the removal of the freed fibre after each stage; that instead of taking off clean fiberized material by the regulation of the suction used, it is better to employ a strong suction that will lift free unfiberized material, re-clean, and if necessary, fiberize the suction product. In this manner the fibre is not broken up by the grinding action of coarse particles of rock.

(7)

THE CONCENTRATION OF THE LOWER GRADE COPPER-NICKEL ORES

C. S. Parsons

The present practice of smelting the copper-nickel ores of the Sudbury district, direct in the blast furnace or by mixing the green ore with a certain portion which had been heap roasted, has given very favourable results, in the past. This practice was especially adaptable to the higher grade ores when consideration is given to the favourable market conditions that have prevailed to within the last year or so.

In the treatment of the lower grade ores of the district, it is a question whether it would not be more economical to apply certain methods of concentration to make a product with a fairly high copper-nickel content prior to smelting operations. By the elimination of the gangue rock and a large portion of the pyrrhotite it was hoped to obtain a product with a copper-nickel content the equal of the blast furnace matte. Such methods of operation would probably eliminate the blast furnace practice in the treatment of the lower grade ores.

The investigations conducted on the lower grade ores were confined to the concentration of the ores from the property of the Burrows Refining Co., to obtain a product containing the metallic minerals, copper, nickel and iron, suited to the refining process of their company, who were carrying on investigations in the manufacture of certain alloy steels; and also, to the concentration of the lower grade ores from other mines of the district by preferential flotation, to determine if the copper-nickel values could not be concentrated into a high grade product, with a good recovery of the values.

The results of the experimental work are given under the following test numbers:—

Test No. 148

(*Preliminary Report.*)

Concentration tests made on nickel ore from the mines of the Burrows Refining Company, in the Sudbury district, Ontario.

Lots No. 1 and No. 2.—The ore which had been crushed to pass 20 mesh before being received at the laboratory, was badly oxidized, and had to be passed through rolls to break up the lumps of cemented particles.

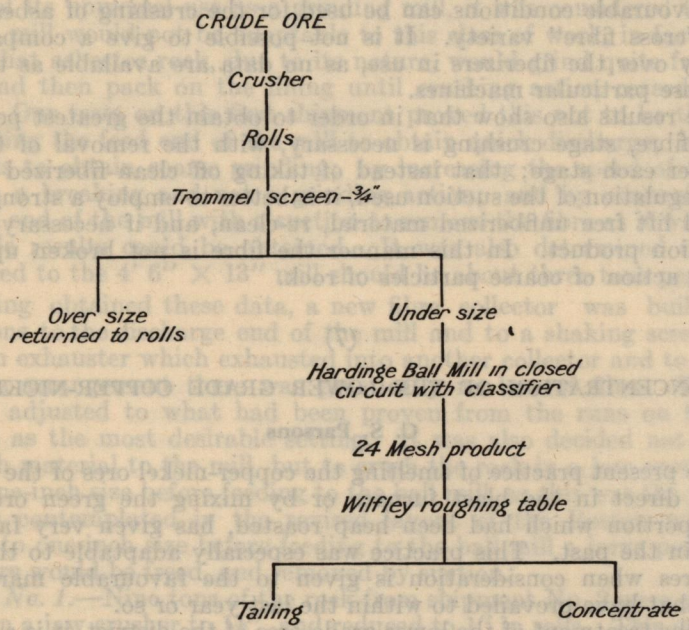


FIG. 3. Flow sheet, concentration test on nickel ore, Test 148, No. 1.

Test No. 1

(See flow sheet, Fig. 3)

Lots 1 and 2 were sampled and fed to a Wilfley table.

Lot 1—	
Weight	1,819 pounds
Analysis	Ni. 2·70% : Cu. 0·25 %
Nickel content	49·1 pounds
Copper content	4·55 pounds
Lot 2—	
Weight	1,290 pounds
Analysis	Ni. 2·62 %
Nickel content	33·8 pounds
Lots 1 and 2 total—	
Weight	3,109 pounds
Analysis	Ni. 2·67 %
Nickel content	83·01 pounds

The ore was fed over a 24 mesh Callow screen, the −24 mesh material going direct to a Wilfley table without further sizing, the +24 mesh material to a storage tank for separate treatment later over the same table.

Results of tabling—

−24 mesh product—	
Table concentrate	−24 mesh
Weight	1,712 pounds
Analysis	3·60% Ni.
Content	61·63 pounds Ni.
Percentage of total nickel values	74·2
Table tailing	−24 mesh
Weight	499 pounds
Analysis	1·15% Ni.
Content	5·74 pounds Ni.
Percentage of total nickel values	6·9
+24 mesh product—	
Table concentrate	+24 mesh
Weight	150 pounds
Analysis	2·0% Ni.
Content	3·0 pounds Ni.
Percentage of total nickel values	3·6
Table tailing	+24 mesh
Weight	393·25 pounds.
Analysis	0·6% Ni.
Content	2·13 pounds Ni.
Percentage of total nickel values	2·6

Summary of the above results—

Product	Weight pounds	Analysis per cent Ni.	Content pounds Ni.	Percentage of total Ni. value
Concentrate	1,862	3·47	64·63	77·8
Tailing	853·75	0·92	7·87	9·5
Loss	395·25	2·67	10·51	12·7
Heads	3,109·00	2·67	83·01	100·0

This large loss was due to the heavy sulphide lying in the pumps and pipe lines, and also to slime loss from tailing. There was also approximately 0·25 per cent soluble nickel in the ore, a part of which went into solution in the mill water. The recovery of 77·8 per cent is very conservative, and it is probable that the actual recovery was over 80 per cent.

The ore tabled very well considering that the feed to the table was not sized. The capacity of the table was, however, low, the rate of feed being 241·5 pounds per hour. It is probable that a different system of riffling would increase the capacity considerably.

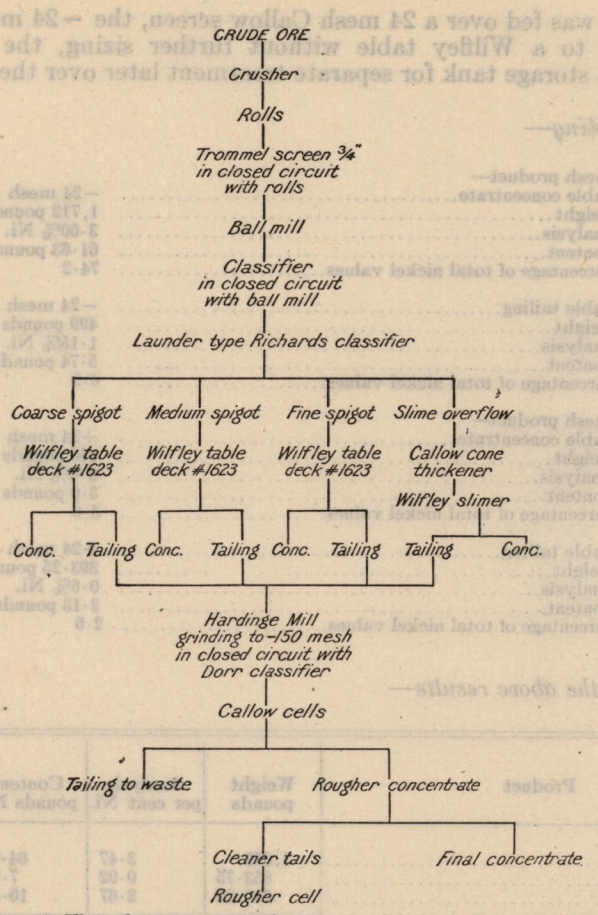


FIG. 4. Flow sheet, concentration test on nickel ore, Test 148, No. 2.

Test No. 2

(See flow sheet, Fig. 4.)

Lot No. 4.—This lot of ore was received in lump form. It was crushed in a jaw crusher to ¾", and then ground in a Hardinge ball mill to pass a 24 mesh Callow belt screen, the oversize being returned to the mill. The undersize from the screen was led to two Richards hydraulic launder type classifiers, where three products were made, a coarse spigot product, a fine spigot product, and an overflow slime. Each of these products was fed separately to a Wilfley table.

Weight of ore fed to mill	11,912 pounds
Analysis	2·50% Ni.
Content	297·8 pounds Ni.
Loss of weight during test	1,060 pounds
Analysis calculated of material lost	2·8% Ni.

The loss of weight was due chiefly to the heavy sulphide remaining in the ball mill and to a loss of slimes from the collecting tank, into which the overflow from the Richards classifier was caught and settled.

Screen Analyses of Feeds to Classifier and Table:

Mesh	Feed to classifier		Feed to table from first spigot		Feed to table from second spigot	
	Weight gms.	%	Weight gms.	%	Weight gms.	%
− 24 + 28	5	0·30	93	3·73	0·4	0·02
− 28 + 35	51	2·20	681	27·31	4·4	0·26
− 35 + 48	53	3·80	685	27·44	133·0	7·74
− 48 + 65	124	0·05	504	20·20	275·0	16·00
− 65 +100	165	12·00	312	12·50	427·0	24·85
−100 +150	134	9·70	95	3·81	266·0	15·47
−150 +200	437	31·90	65	2·60	261·0	15·13
−200	428	31·30	60	2·40	352·0	20·48

Concentration of coarse spigot product.—The product was fed to a Wilfley table with a No. 1623 deck. The table was fed at a rate which produced 0·21 tons of concentrate per hour. An approximate increase of 25 per cent in this capacity could be safely figured, which would give an average capacity of 0·26 tons of concentrate per hour.

Concentrate
Weight.....1031·0 pounds
Analysis.....2·20% Ni.
Content.....22·7 pounds Ni.
Tailing
Weight.....Not known
Analysis.....0·55% Ni.

Concentration of fine spigot product.—This product was fed to a Wilfley table with a No. 1623 deck. The table was fed at a rate which produced 0·13 tons of concentrate per hour. The table, however, was operated at approximately 50 per cent of its capacity, and an estimated maximum capacity would be about 0·26 tons per hour of this material.

Concentrate
Weight.....1,326·5 pounds
Analysis.....2·50% Ni.
Content.....33·15 pounds Ni.
Tailing
Weight.....Not taken
Analysis.....0·45% Ni.

Concentration of the overflow, or slime product from the classifier.—This product was fed to a Wilfley table with a No. 1623 deck, at a rate which produced 0·173 tons, or 346 pounds of concentrate per hour. A 10 per cent increase could be figured on this capacity, giving 0·187 tons of concentrate per hour, but this would be about the maximum capacity of the table.

Concentrate
Weight.....3,809·0 pounds
Analysis.....3·50 % Ni.
Content.....133·50 pounds Ni.
Tailing
Weight.....Not known
Analysis.....1·50% Ni.

Tailings re-ground in pebble mill.—The whole table tailing was re-ground in a Hardinge mill to pass an 80 mesh Callow belt screen, and treated by flotation in Callow cells. The reagents used were soda ash and X-Y mixture.

Concentrate
Weight.....223 pounds
Analysis.....15·5% Ni.
Content.....33·45 pounds Ni.
Tailing
Weight.....Not known
Analysis.....0·80% Ni.

SUMMARY OF PRODUCTS

Product	Weight pounds	Analysis % Ni.	Content pounds	Percentage of total nickel	
				Direct %	Accum. %
Concentrate No. 1	1,031·0	2·20	22·7	8·75	8·75
" " 2	1,326·5	2·50	33·2	12·80	21·55
" " 3	3,809·0	3·50	133·5	51·45	73·00
Callow Con. No. 4	223·0	15·50	34·5	13·30	86·30
Total Conc	6,389·5	3·51	223·96	86·30	86·30
Tailings	4,462·5	0·80	35·70	13·70	.....

This gives a recovery of 86·30 per cent of the nickel with a production of 0·589 tons of concentrate per ton of ore.

CONCLUSIONS

Lot 4, Test 2.—It will be noted that only 73 per cent of the nickel was saved by table concentration, and that this low recovery was due to the large loss in the tails from the tabling of the overflow product from the classifier. This overflow product contained over 60 per cent of the total nickel values, and the tailing from it analysed 1·5 per cent nickel, while the tailing from the spigot products contained only 0·55 per cent and 0·45 per cent nickel respectively. There is no doubt that a better saving could have been effected by operating the table on a lighter feed, but it is very evident that a further classification of the overflow product should be made before attempting to table it. In the proposed flow sheet, Fig. 4, a third classifier has been shown, which would divide this overflow product into a fine sand product, and a slime overflow product, for treatment on separate tables.

A better recovery was made in test No. 1, where the ore was tabled direct without either sizing or classification, but this result was effected by the slow rate of feed to the table. The table treated only 241·5 pounds per hour, or 0·120 tons, a rate of feed so slow as to be practically prohibitive.

Flow sheet, Fig. 5, gives the results that could be expected if classification was dispensed with.

It is also evident that flotation can be applied very successfully to the table tailing after re-grinding, and we have found by more recent tests that the recovery by flotation can be greatly increased.

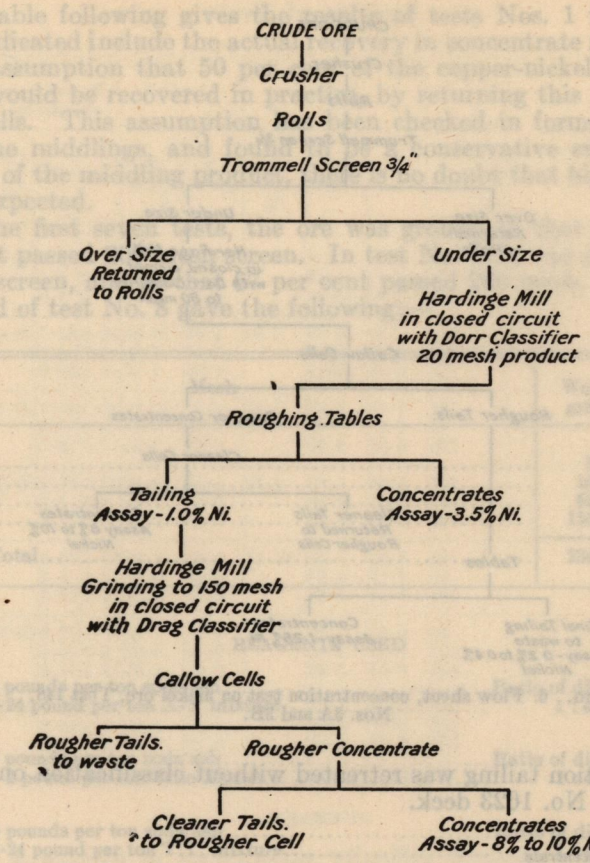


FIG. 5. Flow sheet, concentration test on nickel ore, Test 148, No. 3

Test No. 3A

(See flow sheet, Fig. 6.)

Lot No. 5.—The ore was received in lump form. It was crushed in a jaw crusher to 3/4", and ground in a Hardinge mill to pass an 80 mesh Callow belt screen. The oversize was returned to the ball mill.

Head sample..... 2·60 per cent Ni.

Part of this ore was floated in Callow cells, using soda ash and X-Y reagents.

Flotation concentrate
Weight.....359 pounds
Analysis.....11·3% Ni.
Recovery.....65·6%
Flotation tailing
Weight.....Not taken
Analysis.....1·0% Ni.

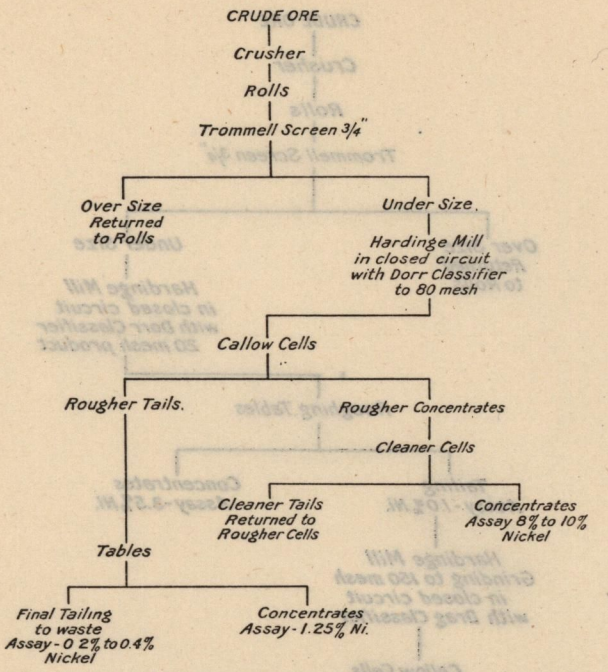


FIG. 6. Flow sheet, concentration test on nickel ore, Test 148, Nos. 3A and 3B.

The flotation tailing was retreated without classification on a Wilfley table, with a No. 1623 deck.

Table concentrate
Weight.....1,198·5 pounds
Analysis.....1·25% Ni.

Table tailing
Weight.....Not taken
Analysis.....0·20% Ni.

SUMMARY

Product	Weight pounds	% of total weight	Analysis % Ni.	Recovery %
Concentrate No. 1	359·0	.....	11·30	.....
" " 2	1,198·5	.....	1·25	.....
Total concentrate	1,557·7	.....	3·50	98·0
Total tailing	.....	.....	0·20	2·0

Recovery of nickel in flotation concentrate..... 65·0%
Recovery of nickel in flotation and table concentrate..... 98·9%
Analysis of nickel in total concentrate..... 3·50%

Test No. 3B

(See flow sheet, Fig. 6)

A further portion of this ore was treated by flotation, using oil, instead of the X-Y reagents, and lime instead of soda ash. Reagents used: lime, and a gravity fuel oil, with a paraffine base.

Flotation concentrate
Weight.....710 pounds
Analysis.....7·35% Ni.
Recovery.....74·5% Ni. content
Flotation tailing
Weight.....Not taken
Analysis.....0·90% Ni.

This tailing was tabled with the following results:—

Table concentrate
Weight.....950 pounds
Analysis.....1·70% Ni.
Table tailing
Weight.....Not taken
Analysis.....0·53% Ni.

SUMMARY

Product	Weight pounds	% of total weight	Analysis % Ni.	Recovery %
Concentrate No. 1	710	.....	7·35	.....
" " 2	950	.....	1·75	.....
Total concentrate	1,660	.....	4·12	91·3
Total tailing	.....	.....	0·53	8·7

Recovery of nickel in total concentrate..... 91·3%
Analysis of nickel in total concentrate..... 4·12%
Tons of concentrate produced per ton crude..... 0·58

CONCLUSIONS

The flotation of this ore in these first two tests was not very successful due to the difficulty in maintaining a froth on the cells. A series of small scale tests have since been made, and this difficulty was found to be due to the ore not being ground fine enough. It was found necessary to grind the ore fine for two reasons; first, to free the nickel-bearing mineral, and second, to produce enough rock and sulphide slimes to stabilize and toughen the froth in the flotation cells, so that it could be brought over.

The method of treatment, outlined in flow sheet, Fig. 6, and described in test No. 3A, seems to be the most practical method of producing the product required by the Burrows Refining Company.

It is evident that by the above process of treatment (see flow sheet Fig. 6), namely that of treating the crude ore by flotation, and then tabling the flotation tailing, great flexibility can be obtained. It would be possible to produce a concentrate which would contain any desired ratio of nickel to iron between the limits of 3·5 per cent nickel and 8·0 per cent nickel, and still average a fair recovery. The copper content of this concentrate would vary with the nickel content, and would range between 0·4 per cent cu. to 1·25 per cent cu.

Some Small Scale Tests by Flotation

The object of these tests was to produce a high grade nickel-copper concentrate by differential flotation.

Test No. 5

One thousand grams of ore was crushed in a small ball mill until 65 per cent passed a 200 mesh screen. The reagents used were 5 pounds soda ash per ton and 1 pound X-Y reagent per ton.

Product	Weight grams	Analysis	Recovery Ni	——
Concentrate	307	6·60%	83·5%	Total recovery, assuming that 60% of the Ni. content in middling could be recovered in the concentrate, 87·7%.
Middling	142	1·30%	.....	
Tailing	538	0·45%	.....	
Total	987			

Test No. 6

One thousand grams of ore was crushed in a small ball mill until 68 per cent passed a 200 mesh screen. The reagents used were 3 pounds of soda ash and 0·6 pounds of X-Y reagent per ton.

Product	Weight grams	Analysis Ni.	Recovery	——
Concentrate	151	11·60%	68·7%	Total recovery, assuming that 60% of the Ni. content in middling could be recovered in the concentrate, 80%.
Middling	201	2·20%	17·5%	
Tailing	660	0·53%	13·7%	
Total	1,012			

CONCLUSIONS

A second series of these tests will be run at a later date, to determine if it is possible by finer crushing to produce a higher grade nickel concentrate with a higher recovery.

These results are very encouraging, and show fairly conclusively that the Sudbury ores can be concentrated by flotation.

Test No. 155

A shipment of 100 pounds of copper-nickel ore was received at the Ore Dressing and Metallurgical Laboratories, October 10, 1921, from the International Nickel Company, Copper Cliff, Ont. The ore was from their No. 3 mine.

The object of the experimental work on this ore was to determine if it could be concentrated by flotation.

A number of small scale tests were made using the Alphabetical reagents of the General Engineering Company. The results of this work are contained in the two following tables.

Table No. I gives the results of tests Nos. 1, 2, 4 and 5. Recoveries indicated include the actual recovery in the concentrate made, together with 50 per cent of the copper-nickel values in the middling, which would be recovered in practice by returning them to the rougher cells. Table No. II gives the results of tests Nos. 6 and 8, in which the middlings were re-run without further grinding. With re-grinding of the middling product, there is no doubt that higher recoveries could be expected.

TABLE I

Test No.	Concentration Products	Weight grams	Analysis			Content		Percentage of values		Total Recoveries		
			Cu. per cent	Ni. per cent	Cu. + Ni. per cent	Cu. gms.	Ni. gms.	Cu.	Ni.	Cu. per cent	Ni. per cent	Cu. + Ni. per cent
1	Concentrate....	106	12·50	9·60	22·10	13·25	10·18	93·2	76·2	95·3	82·4	89·1
	Middling.....	147	0·40	1·13	.....	0·59	1·66	4·1	12·4			
	Tailing.....	758	0·05	0·20	.....	0·38	1·52	2·7	11·3			
2	Concentrate....	75	16·50	10·20	26·70	12·37	7·65	88·3	58·7	91·9	68·8	80·8
	Middling.....	155	0·65	1·70	.....	1·01	2·63	7·2	20·2			
	Tailing.....	785	0·08	0·35	.....	0·63	2·75	4·5	21·1			
4	Concentrate....	95	13·66	9·75	23·41	12·98	9·26	93·6	69·8	95·4	78·3	87·1
	Middling.....	146	0·35	1·55	.....	0·51	2·26	3·7	17·0			
	Tailing.....	758	0·05	0·23	.....	0·38	1·74	2·7	13·1			
5	Concentrate....	170	7·40	6·20	13·60	12·58	10·54	92·0	80·0	93·9	84·5	89·2
	Middling.....	94	0·55	1·27	.....	0·52	1·19	3·8	9·0			
	Tailing.....	730	0·08	0·20	.....	0·58	1·46	4·2	11·0			

Reagents used—

Test No. 1—6 lbs. soda ash, 0·24 lb. X-Y per ton.
" 2—5 lbs. lime, 0·24 lb. X-Y per ton.
" 4—2 lbs. lime, 3 lbs. soda ash, 0·24 lb. X-Y per ton.
" 5—5 lb. soda ash, 0·24 lb. X-Y per ton.

TABLE II

Test No.	Concentration Products	Weight grams.	Analysis			Content		Percentage of values		
			Cu. per cent	Ni. per cent	Cu. + Ni. per cent	Cu. gms.	Ni. gms.	Cu.	Ni.	Cu. + Ni.
6	Concentrate.....	376	7·60	5·70	13·30	28·58	21·43	91·4	86·5	89·2
	Tailing.....	1,594	0·17	0·21	0·38	2·71	3·35	8·6	13·5	10·8
	Ore.....	.....	1·56	1·24	2·80	31·29	24·78	100·0	100·0	100·0
8	Concentrate.....	292	9·10	7·50	16·60	26·57	21·90	96·7	83·4	90·2
	Tailing.....	1,675	0·055	0·26	0·31	0·92	4·35	3·3	16·6	9·8
	Ore.....	.....	1·37	1·31	2·68	27·49	26·25	100·0	100·0	100·0

Reagents used—

Test No. 6—5 lbs. soda ash, 0·2 lb. X-Y per ton
Test No. 8—5 lbs. soda ash, 0·2 lb. X-Y per ton.

NOTE.—T.T. Reagent gives similar results.

Observations and procedure followed in conducting tests.—There is no difficulty in making a fairly high grade copper-nickel concentrate with a good recovery of the copper-nickel values in the ore. The Alphabetical reagents were used, being the most suitable for this class of ore. Further work will be done using other reagents, to determine whether similar, or better results can be obtained.

In using the Alphabetical reagents, the essential point is to carry an alkaline pulp, obtained by the addition of lime, soda ash, or caustic soda. Lime has a tendency to prevent some of the nickel from floating, but gives a higher grade product. Soda ash gives the best recoveries, but a lower grade product. A combination of the two seems to give the best results. The chalcopyrite floats with remarkable ease. With soda ash alone, without any other flotation reagent, a large proportion of it is floated.

No test work was done to determine the degree of fineness to which the ore should be ground to obtain the best results. For the above tests it was ground so that approximately 95 per cent passed a 200 mesh screen. The procedure was as follows: Ore crushed to 20 mesh; 1,000 grams were ground wet in a small ball mill, with the lime or soda ash required, for about 40 minutes; a rougher concentrate and tailing were first made; the rougher concentrate was re-cleaned, producing a concentrate and a middling. In tests Nos. 6 and 8, two 1,000 gram lots were used following the above procedure, in order to obtain sufficient middling for re-floating. The concentrate from re-floating the middling was put with the final concentrate, and the tailing with the rougher tailing, so as to make only two products, concentrate and tailing.

Test No. 160

A shipment of 100 pounds of copper-nickel ore was received at the Ore Dressing and Metallurgical Laboratories, October 20, 1921, and a second shipment, November 15, 1921, from the British America Nickel Corporation, Ltd., Nickelton, Ont.

The object of the experimental work on this ore was to determine if it could be concentrated by flotation.

A number of small scale tests were made using the Alphabetical reagents of the General Engineering Company.

The first shipment of ore received consisted of fines, and was not amenable to concentration by flotation methods. After a number of experiments had been tried, the cause of the failure of the nickel to float was found to be due to oxidation. It was found from previous tests, on other nickel ores, that it is necessary to have freshly broken unoxidized ore to obtain a good recovery of the nickel values by flotation. The nickel-bearing mineral oxidizes very rapidly, and when slightly oxidized does not float readily.

The second shipment consisted of coarse lump ore, recently mined, and was not subject to oxidation to the extent that was the case in the first shipment of fines. It was on this second shipment that the results given in the following table were obtained. This ore gave an analysis as follows:—

Copper	(Cu.)	0·47%
Nickel	(Ni.)	1·17%
Iron	(Fe.)	20·90%
Sulphur	(S.)	11·19%

The table following gives the results of tests Nos. 1 to 8. The recoveries indicated include the actual recovery in concentrate made, together with the assumption that 50 per cent of the copper-nickel values in the middling would be recovered in practice, by returning this product to the rougher cells. This assumption has been checked in former tests by retreating the middlings, and found to be a conservative estimate. With regrinding of the middling product, there is no doubt that higher recoveries could be expected.

For the first seven tests, the ore was ground so that approximately 90 per cent passed 200 mesh screen. In test No. 8, it was crushed to pass 100 mesh screen, and so that 67 per cent passed 200 mesh. A screen test on the feed of test No. 8 gave the following:—

Mesh	Weight grams	Per cent
+100	2·0	0·8
+150	14·0	6·0
+200	62·0	26·5
−200	156·0	66·7
Total	234·0	100·0

REAGENTS USED

Test No. 1—5 pounds per ton soda ash.....Ratio of dilution
0·24 pound per ton X-Y mixture..... 1 : 4

Test No. 2—5 pounds per ton soda ash.....Ratio of dilution
0·2 pound per ton Thio Fizzan..... 1 : 4

Test No. 3—5 pounds per ton soda ash.....Ratio of dilution
0·24 pound per ton T.T. mixture..... 1 : 4

Test No. 4—5 pounds per ton soda ash.....Ratio of dilution
1 pound per ton No. 34 gravity fuel oil..... 1 : 4
0·2 pound per ton No. 5 pine oil.....

Test No. 5—5 pounds per ton soda ash.....Ratio of dilution
0·24 pound per ton X-Y mixture..... 1 : 8

Test No. 6—4 pounds per ton soda ash
1 pound per ton lime
0·2 pound per ton X-Y mixture

Test No. 8—5 pounds per ton soda ash
1 pound per ton No. 34 gravity fuel oil (G.W. Oil Co.)
and No. 5 pine oil (G.N.S.)

Results of Tests

Test No.	Concentration Products	Weight grams	Analysis			Content		Percentage of values			Total recoveries		
			Cu. %	Ni. %	Cu. & Ni. %	Cu. gms.	Ni. gms.	Cu.	Ni.	Cu. & Ni.	Cu. %	Ni. %	Cu. & Ni. %
1	Concentrate	155	2·6	6·2	8·8	4·03	9·61	87·4	82·6	84·0	90·0	86·8	87·8
	Middling	159	0·15	0·62	.....	0·24	0·98	5·2	8·4				
	Tailing	694	0·05	0·15		0·34	1·04	7·4	8·9				
2	Concentrate	175	2·30	5·75	8·05	4·03	10·06	89·3	84·7	86·0	90·7	87·0	88·0
	Middling	118	0·10	0·45		0·12	0·53	2·7	4·5				
	Tailing	715	0·05	0·18		0·36	1·29	8·0	10·8				
3	Concentrate	194	2·15	5·26	7·41	4·17	10·20	89·7	87·1	82·4	90·9	81·6	84·1
	Middling	82	0·15	0·57		0·12	0·47	2·6	3·6				
	Tailing	730	0·05	0·29		0·36	2·12	7·7	10·6				
4	Concentrate	220	1·90	4·80	6·70	4·18	10·56	91·5	88·5	89·3	92·3	90·3	90·8
	Middling	144	0·05	0·30		0·07	0·43	1·5	3·6				
	Tailing	630	0·05	0·15		0·32	0·94	7·0	7·9				
5	Concentrate	64	3·05	7·12	10·17	1·95	4·56	83·7	76·6	78·6	86·3	80·0	81·8
	Middling	69	0·17	0·56		0·12	0·39	5·1	6·5				
	Tailing	369	0·07	0·27		0·26	1·00	11·2	16·8				
6	Concentrate	66	6·65	13·36	20·01	4·39	8·82	82·1	73·0	75·7	84·9	79·6	81·2
	Middling	122	0·25	1·33		0·30	1·62	5·6	13·4				
	Tailing	825	0·08	0·20		0·66	1·65	12·3	13·6				
8	Concentrate	168	2·70	5·83	8·53	4·53	9·97	93·2	80·4	84·0	94·6	85·5	88·1
	Middling	205	0·07	0·61		0·14	1·25	2·9	10·3				
	Tailing	632	0·03	0·18		0·19	1·14	3·9	9·3				

SUMMARY

There is no apparent difficulty experienced in making a fairly high grade copper-nickel concentrate, with a good recovery of the copper-nickel values in the ore. The Alphabetical reagents were used being the most suitable for this class of ore.

In using these reagents, the essential point is to carry an alkaline pulp, obtained by the addition of lime, soda ash, or caustic soda. Lime has a tendency to prevent some of the nickel from floating, but gives the higher grade product. Soda ash gives the best recoveries, but a lower grade product. A combination of the two seems to give the best results. The chalcopyrite floats with remarkable ease. A large proportion of it is floated with soda ash alone, without any other flotation reagent.

(8)

THE CONCENTRATION OF GARNET FROM ITS ROCK

R. K. Carnochan

There are considerable areas in Canada in which the rock formation is favourable to the presence of garnet and other non-metallic minerals.

Under favourable conditions where deposits of these minerals are close to rail or water transportation and where they are of suitable grade and extent, they could be produced at a profit. Up to the present there has been no production of garnet, but within the last year there has been more or less activity in the way of exploration work on several deposits to determine their extent, grade, and the commercial possibilities of establishing an industry in this country.

Several shipments were received at the Ore Dressing and Metallurgical Laboratories. Two were from the vicinity of Bancroft, Ontario, and one from Depot Harbour, Ontario. The experimental work consisted of determining the garnet content, percentage of recoverable garnet, and methods of concentration and separation of the garnet from the gangue. On account of its favourable situation from a shipping standpoint for the disposal of the concentration products, the work of the shipment from Depot Harbour was more exhaustive, covering more detail than in the case of the other shipments.

Test No. 150 (Part 1)

A shipment of garnet rock weighing 95 pounds was received at the Ore Dressing and Metallurgical Laboratories, April 11, 1921, from Mr. Robt. A. Bryce, Toronto, Ont.

The rock came from the vicinity of Bancroft, Ont., and consisted of garnets of fair size in mica schist.

It was desired that tests be conducted on this rock to produce a high grade garnet concentrate, on which the owner wished to conduct abrasive tests.

A specimen was selected from the sample, and the balance was then crushed in a small jaw crusher to pass 20 mesh, and screened on 40 and 100 mesh screens, with the following results:—

	lbs.	p.c.
−20+ 40	28	30·4
−40+100	37·5	40·8
−100	26·25	28·5
Loss	0·25	0·3
Heads	92	100·0

A small portion was cut out of each of the three sizes and tabled on a small Wilfley table, with the following results:—

−20 +40	Concentrates	159 gms.	3·2% by wt. of heads
	Middlings	306 "	6·1% " "
	Tails	1,052 "	20·9% " "
	Loss	13 "	0·2% " "
	Feed	1,530 "	30·4% " "
−40 +100	Concentrates	127 gms.	2·8% by wt. of heads
	Middlings	97 "	2·2% " "
	Tails	1,590 "	35·3% " "
	Loss	21 "	0·5% " "
	Feed	1,835 "	40·8% " "
−100	Concentrates	82 gms.	1·7% by wt. of heads
	Middlings	90 "	1·8% " "
	Tails	1,147 "	23·0% " "
	Loss	100 "	2·0% " "
	Feed	1,419 "	28·5% " "

A larger portion was then cut out of each of the three sizes and concentrated on a large Wilfley table, with the following results:—

−20 +40	Concentrates... 2 lbs. 2 ozs.	3·1%	by wt. of heads
	Tails.....16 " — "	23·4%	" "
	Loss..... 2 " 10 "	3·9%	" "
	Feed.....20 " 12 "	30·4%	" "
−40 +100	Concentrates... 2 lbs. 5 ozs.	3·3%	by wt. of heads
	Tails.....25 " 8 "	36·5%	" "
	Loss..... 11 "	1·0%	" "
	Feed.....28 " 8 "	40·8%	" "
−100	Concentrates... 1 lbs. — ozs.	1·5%	by wt. of heads
	Tails.....12 " — "	17·5%	" "
	Loss..... 6 " 9 "	9·5%	" "
	Feed.....19 " 9 "	28·5%	" "

CONCLUSIONS

A clean separation has been made of the garnet from the gangue. The results of the above tests show that 7·9 per cent of the weight of the rock is recoverable as garnet concentrate, that is, for every 100 tons of rock treated 7·9 tons of garnet concentrates would be produced.

Tests were run on the magnetic separator and on the electrostatic separator, but a poor separation in each case was obtained. The results are not worthy of recording.

From the results obtained, a satisfactory method of dressing this rock would be, crushing wet to 10 mesh, or possibly finer, classification, and tabling of the classified material.

Test No. 150 (Part 2)

A shipment of garnet rock weighing 95·5 pounds was received at the Ore Dressing and Metallurgical Laboratories, May 7, 1921, from Mr. Robt. A. Bryce, Toronto, Ont.

The rock was from the vicinity of Bancroft, Ont., and consisted of garnets of fair size in mica schist.

It was desired that tests be conducted on this rock to produce a high grade garnet concentrate, on which the owner wished to conduct abrasive tests.

A specimen was selected from the sample, and the balance was then crushed in a small jaw crusher to pass 20 mesh, and screened on 40 and 100 mesh screens, with the following results:—

	lbs.	p.c.
−20+ 40	48	52·0
−40+100	29·5	32·0
−100	14·5	15·7
Loss	0·25	0·3
Heads	92·25	100·0

The −20 + 40, and the −40 +100, sizes were tabled separately on a large Wilfley table, and the −100 material on a small Wilfley table, with the following results:—

Size	Product	Weight pounds	Percentage by weight of heads
−20 +40	Concentrate	11·80	12·8
	Tails	33·00	35·7
	Loss	3·20	3·5
	Feed	48·00	52·0
−40 +100	Concentrate	7·74	8·4
	Tails	20·00	21·7
	Loss	1·76	1·9
	Feed	29·50	32·0
−100	Concentrate	2·64	2·9
	Tails	10·25	11·1
	Loss	1·61	1·7
	Feed	14·50	15·7

CONCLUSIONS

A clean separation has been made of the garnet from the gangue. The results of the above tests show that 24·1 per cent of the weight of the rock is recoverable as garnet concentrates, that is, for every 100 tons of rock treated, 24·1 tons of garnet concentrates would be produced.

From the results obtained, a satisfactory method of dressing this rock would be, crushing wet to 10 mesh, or possibly finer, classification, and tabling of the classified material.

Test No. 158

Source of shipment.—A shipment of 127 bags of garnet rock, weight 13,528 pounds, was received at the Ore Dressing and Metallurgical Laboratories, November 22, 1921, from a garnet deposit at Depot Harbour, Ont. The deposit is situated on railway ground on the harbour, with excellent shipping facilities for disposal of the products. The shipment was made by Robt. A. Bryce, Toronto.

Exploration work.—The deposit was stripped at certain intervals and trenched. Moiled samples were cut from the bottom of the trenches. Some twelve trenches were sampled in this manner, and each bag represented a sample over a certain width along the trench. The trenches sampled were marked 1 to 12, and starting at one end of a trench with sample A for a certain length, followed by sample B, and so on in succession along the trench and across the deposit until the end of the trench was reached. The stripping and trenching showed the width and extent of the deposit, and determined to some extent the workable area from which the rock could be extracted for the recovery of the garnet content.

Purpose of the shipment.—The samples as submitted were for the purpose of determining the garnet content of the deposit as far as exploration work was conducted in the way of stripping and trenching over the area; to determine the average content of recoverable garnet; and also for determining methods of recovering the garnet from the rock in grades suitable for the market.

Characteristics of the deposits.—The deposits cover considerable area, as shown on the plan, in places over 200 feet wide, separated by bodies of gneiss. They occur in the gneiss and are composed of garnets up to one-half inch in size in a gneissic mica schist. The rock showed that it contained approximately 15 per cent garnet, 55 per cent mica, and 30 per cent silica and other gangue minerals. From tests made for the owners, the garnet had high abrasive qualities, and was suitable for the trade.

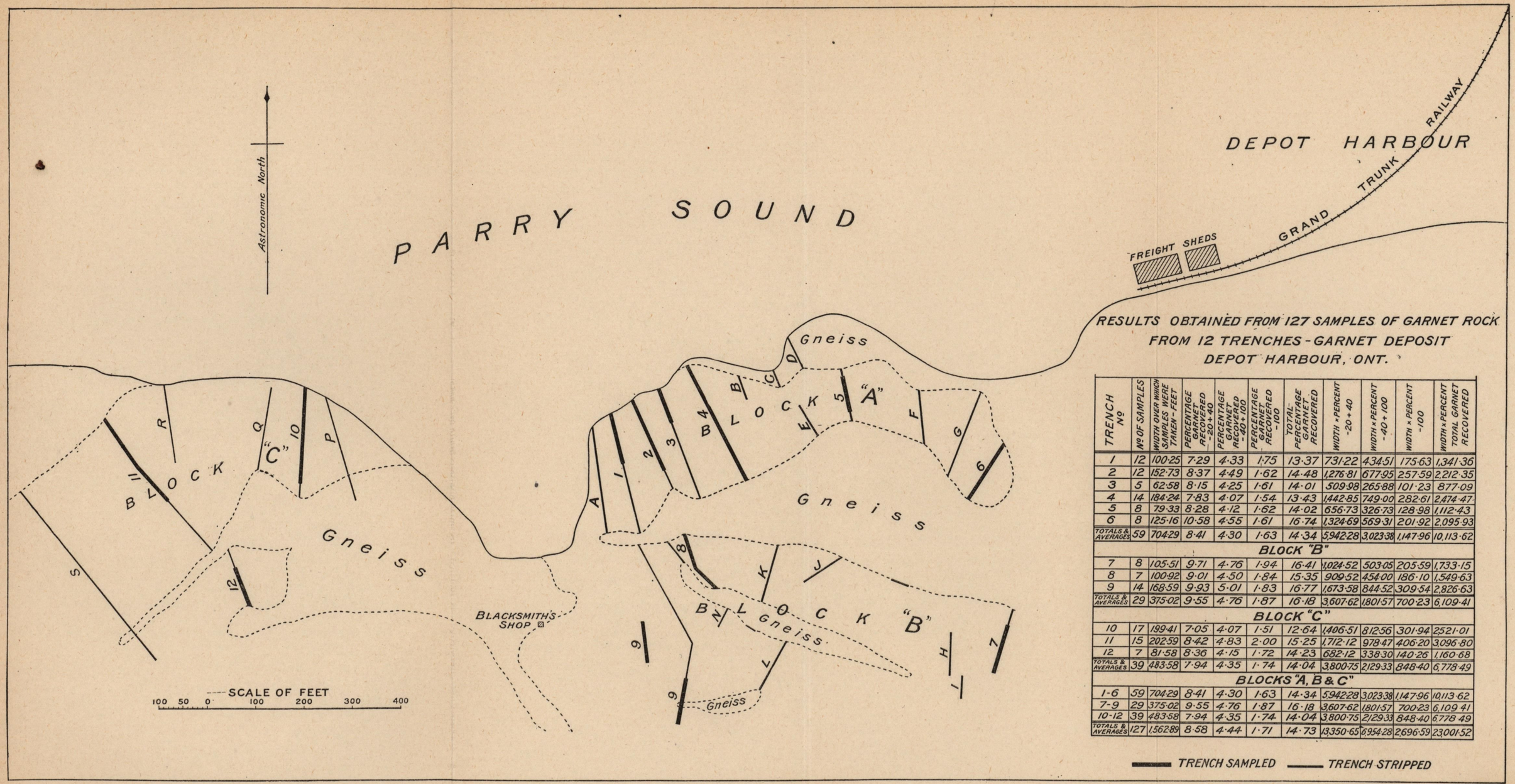
Plan of deposits.—The accompanying plan shows the location of the deposits; their extent as far as exploration work has proven them; the stripping and trenches; those trenches which have been sampled for the determination of garnet content and for material for experimental work on methods of treatment for the recovery of the garnet; and also tables showing the percentage of recoverable garnet. (See Fig. 7.)

Examination of garnet used for the manufacture of garnet paper.—The greatest portion of the garnet consumption is used for this purpose. It was, therefore, necessary to obtain some idea as to the size of the particles used in the manufacture of the various grades of paper. Samples were secured which gave the following screen analyses:—

Grade No.	Per cent on mesh—Tyler standard screen Sc.						
	+20	+28	+35	+48	+65	+100	−100
2½	0·0	12·7	86·6	0·3	0·4	.....	.....
2	0·0	0·4	98·3	0·8	0·4	0·2	.....
1½	0·0	0·0	52·2	46·1	1·4	0·3	.....
1	0·0	0·0	0·0	83·5	16·3	0·1	0·1
½	0·0	0·0	0·0	3·0	87·5	9·1	0·4

This test showed that the largest percentage of garnet used for the manufacture of garnet paper was all through 20 mesh (Tyler Standard) and practically all on 100 mesh, and as far as this particular trade was concerned there was no advantage in providing a product coarser than 20 mesh, especially when a cleaner product could be obtained around this mesh, as any attached or included gangue in the garnet crystal would be released, and treatment on jigs could be eliminated, provided that the reduction of the rock to this mesh did not result in an excessive amount of fine garnet −100 mesh.

From data collected it is found that the grades of garnet produced are known to the trade as Nos. 5, 4½, 4, 3½, 3, 2½, 2, 1½, 1, ½, 0, 2/0, 3/0 and 7/0. This means a considerably wider range than that given in the above table, and probably from 3 mm. size to 200 mesh. The greatest consumption is no doubt between 20 and 200 mesh. An examination of the garnet crystals from this deposit shows that the crystals are more or



RESULTS OBTAINED FROM 127 SAMPLES OF GARNET ROCK FROM 12 TRENCHES - GARNET DEPOSIT DEPOT HARBOUR, ONT.

TRENCH No	No OF SAMPLES	WIDTH OVER WHICH SAMPLES WERE TAKEN - FEET	PERCENTAGE GARNET RECOVERED -20+40	PERCENTAGE GARNET RECOVERED -40+100	PERCENTAGE GARNET RECOVERED -100	TOTAL PERCENTAGE GARNET RECOVERED	WIDTH × PERCENT -20+40	WIDTH × PERCENT -40+100	WIDTH × PERCENT -100	WIDTH × PERCENT TOTAL GARNET RECOVERED
1	12	100·25	7·29	4·33	1·75	13·37	731·22	434·51	175·63	1,341·36
2	12	152·73	8·37	4·49	1·62	14·48	1,276·81	677·95	257·59	2,212·35
3	5	62·58	8·15	4·25	1·61	14·01	509·98	265·88	101·23	877·09
4	14	184·24	7·83	4·07	1·54	13·43	1,442·85	749·00	282·61	2,474·47
5	8	79·33	8·28	4·12	1·62	14·02	656·73	326·73	128·98	1,112·43
6	8	125·16	10·58	4·55	1·61	16·74	1,324·69	569·31	201·92	2,095·93
TOTALS & AVERAGES	59	704·29	8·41	4·30	1·63	14·34	5,942·28	3,023·38	1,147·96	10,113·62
BLOCK "B"										
7	8	105·51	9·71	4·76	1·94	16·41	1,024·52	503·05	205·59	1,733·15
8	7	100·92	9·01	4·50	1·84	15·35	909·52	454·00	186·10	1,549·63
9	14	168·59	9·93	5·01	1·83	16·77	1,673·58	844·52	309·54	2,826·63
TOTALS & AVERAGES	29	375·02	9·55	4·76	1·87	16·18	3,607·62	1,801·57	700·23	6,109·41
BLOCK "C"										
10	17	199·41	7·05	4·07	1·51	12·64	1,406·51	812·56	301·94	2,521·01
11	15	202·59	8·42	4·83	2·00	15·25	1,712·12	978·47	406·20	3,096·80
12	7	81·58	8·36	4·15	1·72	14·23	682·12	338·30	140·26	1,160·68
TOTALS & AVERAGES	39	483·58	7·94	4·35	1·74	14·04	3,800·75	2,129·33	848·40	6,778·49
BLOCKS "A, B & C"										
1-6	59	704·29	8·41	4·30	1·63	14·34	5,942·28	3,023·38	1,147·96	10,113·62
7-9	29	375·02	9·55	4·76	1·87	16·18	3,607·62	1,801·57	700·23	6,109·41
10-12	39	483·58	7·94	4·35	1·74	14·04	3,800·75	2,129·33	848·40	6,778·49
TOTALS & AVERAGES	127	1,562·89	8·58	4·44	1·71	14·73	13,350·65	6,954·28	2,696·59	23,001·52

TRENCH SAMPLED — TRENCH STRIPPED

FIG. 7. Plan of Garnet deposit, Parry Island, Parry Sound district, Ont.

less fractured, especially on the outside, and in order to produce individual particles containing no fractures, reduction to 20 mesh is necessary. There is, therefore, no particular advantage in making a jig product, as this product would probably have to be crushed in order to obtain particles free from fractures.

Examination of garnet shipped from the trade.—Two samples of garnet were secured, one of coarse material and the other of finer material. Screen analyses were made on both these products. Tyler standard screens were used.

Mesh	Coarse sample	Fine sample
	%	%
+6	0·0	.....
+8	7·1	.....
+10	39·3	.....
+14	30·7	.....
+20	14·6	.....
+28	4·8	.....
+35	2·8	.....
+48	0·4	0·0
+65	0·1	1·6
+100	0·1	35·3
+150	0·1	32·2
+200	.....	20·7
−200	.....	10·2

The screen tests on these two products show that coarser material than 20 mesh is shipped for the trade, and material finer than 100 mesh is used. It may be that the excess of coarse garnet is ground to the required mesh after being concentrated as coarse as possible on jigs. An examination of the coarser particles showed them to contain included gangue in the fractures. The finer material may be used for the manufacture of the finer grades of garnet paper. Whether it will be profitable to save the fine garnet is a matter that will be determined in the milling practice. The table duty for the recovery of the fines is low, so that it may be more profitable to increase the rock tonnage to the mill and recover coarse garnet, eliminating the fines with the tailings.

Procedure followed for determining percentage of recoverable garnet.—Each bag containing the sample was weighed and the contents crushed in a small jaw crusher to about one-half inch, and a sample cut out by repeated passes through a Jones riffled sampler until a sample of from five to ten pounds was obtained for the small test. The same number of passes was given each sample, so that the remaining large portion would bear the same relation to the whole as before the samples were taken. This small sample was then reduced to 20 mesh by stage crushing in a small set of rolls, so as to make as little fines as possible.

The sample through 20 mesh was sized on 40 and 100 mesh, giving three sizes, −20+40, −40 +100 and −100. Each of the three sizes was run separately over a small laboratory type Wilfley concentrator, making a concentrate, a middling, and a tailing. The middling product was re-run in each case, making a concentrate and tailing. The concentrate was

added to the concentrate from the first pass over the table, and the tailing to the tailing from the first pass, so that only a final concentrate and tailing resulted from the operations. In this manner the percentage of recoverable garnet from each size was obtained, as well as the total percentage. The results are given in tables following under the heading "Results of small scale test work on individual samples."

After completion of the above small scale work, each concentrate—−20+40, −40+100 and −100—from every sample was cut once in a small Jones riffled sampler, one half was put back in the respective sample containers, and the other half was combined, making up three concentrate products, −20+40, −40+100 and −100, each representing a sample of one of the three sizes from all the samples submitted. Each of these three sizes was run over a magnetic separator to determine the percentage of gangue still remaining in the various sizes so that a correction could be made on the results of the laboratory table work. This is given under the heading "Magnetic Separation Tests on laboratory concentrator products."

Results of Small Scale Test Work on Individual Samples:—

Results obtained from 127 moiled samples of Garnet Rock from 12 trenches: Garnet Deposit, Depot Harbour

Trench No.	Sample No.	Width of sample, feet	Percentage garnet recovered				Percentage × width			
			−20+40	−40+100	−100	Total	−20+40	−40+100	−100	Total
1.....	A	14·00	6·95	5·29	2·22	14·46	97·30	74·06	31·08	202 44
	B	8·67	7·59	4·03	1·57	13·19	65·81	34·94	13·61	114·36
	C	6·00	7·60	4·06	1·60	13·26	45·60	24·36	9·60	79·56
	D	5·33	7·32	4·40	1·48	13·20	39·02	23·45	7·89	70·36
	E	8·42	8·76	4·66	1·97	15·39	73·76	39·24	16·58	129·58
	F	5·00	6·73	3·43	1·43	11·59	33·65	17·15	7·15	57·95
	G	11·00	6·86	4·59	1·46	12·91	75·46	50·49	16·06	142·01
	H	8·00	6·79	3·61	1·31	11·71	54·32	28·88	10·48	93·68
	I	8·58	7·86	3·33	1·65	12·84	67·44	28·57	14·16	110·17
	J	9·25	6·44	4·39	1·58	12·41	59·58	40·59	14·62	114·79
	K	6·00	6·78	4·68	2·10	13·56	40·68	28·08	12·60	81·36
	L	10·00	7·86	4·47	2·18	14·51	78·60	44·70	21·80	145·10
Totals and averages	12	100·25	7·29	4·33	1·75	13·37	731·22	434·51	175·63	1,341·36
2.....	A	14·00	8·18	4·21	1·62	14·01	114·52	58·94	22·68	196·14
	B	19·00	9·06	3·99	1·66	14·71	172·14	75·81	31·54	279·49
	C	12·33	7·20	3·79	1·41	12·40	88·78	46·73	17·38	152·89
	D	12·75	7·37	4·67	1·38	13·42	93·96	59·54	17·60	171·10
	E	13·33	7·95	3·70	1·39	13·04	105·97	49·32	18·53	173·82
	F	12·33	6·52	4·51	1·21	12·24	80·39	55·61	14·92	150·92
	G	12·50	9·82	3·67	1·71	15·20	122·75	45·87	21·38	190·00
	H	12·67	6·68	3·75	1·50	11·93	84·64	47·51	19·00	151·15
	I	12·33	11·34	4·42	2·01	17·77	139·82	54·50	24·78	219·10
	J	12·83	8·22	5·76	1·88	15·86	105·46	73·90	24·12	203·48
	K	12·83	8·98	5·76	2·00	16·74	115·21	73·90	25·66	214·77
	L	5·83	9·12	62·3	3·43	18·78	53·17	36·32	20·00	109·49
Totals and averages	12	152·73	8·37	4·49	1·62	14·48	1,276·81	677·95	257·59	2,212·35
3.....	A	8·01	8·01	7·28	2·34	17·63	64·08	58·24	18·72	141·04
	B	9·00	9·98	5·91	2·24	18·13	89·82	53·19	20·16	163·17
	C	14·00	10·13	4·29	1·39	15·81	141·82	60·06	19·46	221·34
	D	15·58	7·56	3·06	1·51	12·13	117·78	47·67	23·53	188·98
	E	16·00	6·03	2·92	1·21	10·16	96·48	46·72	19·36	162·56
Totals and averages	5	62·58	8·15	4·25	1·61	14·01	509·88	265·88	101·23	877·09
4.....	A	10·42	7·03	4·26	1·47	12·76	73·25	44·39	15·32	132·96
	B	15·92	8·24	3·33	1·45	13·02	131·18	53·01	23·08	207·28
	C	15·83	7·30	4·75	1·85	13·90	115·56	75·19	29·29	220·04
	D	17·00	10·79	4·47	1·68	16·94	183·43	75·99	28·56	287·98
	E	15·00	7·06	4·07	1·16	12·29	105·90	61·05	17·40	184·35
	F	15·50	6·19	3·34	1·42	10·95	95·95	51·77	22·01	169·73
	G	15·83	6·29	3·25	1·58	11·12	99·57	51·45	25·01	176·03
	H	14·00	7·42	4·26	1·73	13·41	103·88	59·64	24·22	187·74
	I	12·58	5·95	5·16	1·46	12·57	74·85	64·91	18·37	158·13
	J	12·83	8·33	4·81	1·68	14·82	106·87	61·71	21·55	190·14
	K	11·83	7·93	3·85	1·30	13·08	93·81	45·55	15·38	154·73
	L	8·50	9·67	3·78	1·34	14·79	82·20	32·12	11·39	125·71
	M	13·50	9·60	3·94	1·65	15·19	129·59	53·19	22·28	205·07
	N	5·50	8·51	3·46	1·59	13·56	46·81	19·03	8·75	74·58
Totals and averages	14	184·24	7·83	4·07	1·54	13·43	1,442·85	749·00	282·61	2,474·47

Results obtained from 127 moiled samples of Garnet Rock from 12 trenches: Garnet Deposit, Depot Harbour—Continued.

Trench No.	Sample No.	Width of sample, feet	Percentage garnet recovered				Percentage × width			
			−20+40	−40+100	−100	Total	−20+40	−40+100	−100	Total
5.....	A	11·00	7·10	2·55	1·28	10·93	78·10	28·05	14·08	120·23
	B	9·67	10·84	4·05	1·61	16·50	104·82	39·16	15·57	159·55
	C	7·92	7·61	4·27	1·80	13·68	60·27	33·82	14·26	108·35
	D	10·25	9·44	5·13	1·84	16·41	96·76	52·58	18·86	168·20
	E	9·83	8·49	3·72	1·40	13·61	83·46	36·57	13·76	133·79
	F	10·83	8·39	3·49	1·64	13·52	90·86	37·80	17·76	146·42
	G	13·50	7·21	5·27	1·73	14·21	97·33	71·15	23·36	191·84
	H	6·33	7·13	4·36	1·79	13·28	45·13	27·60	11·33	84·06
Totals and averages	8	79·33	8·28	4·12	1·62	14·02	656·73	326·73	128·98	1,112·43
6.....	A	15·25	8·32	4·24	1·90	14·46	126·86	64·65	28·98	220·51
	B	13·83	8·43	8·55	0·98	17·96	116·59	118·25	13·55	248·39
	C	15·50	6·20	3·52	1·24	10·96	96·10	54·56	19·22	169·88
	D	15·00	7·62	2·98	1·17	11·77	114·30	44·70	17·55	176·55
	E	15·25	6·18	3·13	1·23	10·54	94·24	47·73	18·76	160·74
	F	17·83	12·09	4·70	1·08	17·87	215·56	83·80	19·26	318·62
	G	17·00	17·11	2·47	2·77	22·35	290·87	42·00	47·09	379·95
	H	15·50	17·43	7·33	2·42	27·18	270·17	113·62	37·51	421·29
Totals and averages	8	125·16	10·58	4·55	1·61	16·74	1,324·69	569·31	201·92	2,095·93
7.....	A	12·42	10·24	4·76	1·85	16·85	127·18	59·12	22·98	209·28
	B	10·00	10·50	4·82	2·12	17·44	105·00	48·20	21·20	174·40
	C	11·42	11·44	5·29	1·59	18·32	130·64	60·41	18·16	209·21
	D	13·00	10·12	4·29	1·61	16·02	131·56	55·77	20·93	208·26
	E	13·50	9·97	4·64	1·98	16·59	134·60	62·64	26·73	223·96
	F	12·50	8·62	4·70	2·30	15·62	107·75	58·75	28·75	195·25
	G	15·67	8·96	5·45	2·28	16·69	140·40	85·40	35·73	261·53
	H	17·00	8·67	4·28	1·83	14·78	147·39	72·76	31·11	251·26
Totals and averages	8	105·51	9·71	4·76	1·94	16·41	1,024·52	503·05	205·59	1,733·15
8.....	A	16·42	9·13	4·10	1·51	14·74	149·90	67·32	24·79	242·02
	B	13·00	8·89	5·29	1·95	16·13	115·56	68·77	25·35	209·69
	C	13·67	8·54	4·03	1·68	14·25	116·74	55·09	22·97	194·80
	D	13·17	11·18	4·69	2·18	18·05	137·24	71·77	28·71	237·72
	E	15·58	8·25	4·10	1·83	14·18	128·54	63·88	28·51	220·92
	F	16·33	7·76	3·67	1·64	13·07	126·72	59·93	26·79	213·43
	G	15·75	8·56	4·27	1·84	14·67	134·82	67·24	28·98	231·05
Totals and averages	7	100·92	9·01	4·50	1·84	15·35	909·56	454·00	186·10	1,549·63
9.....	A	13·00	11·59	5·62	1·52	18·73	150·67	73·06	19·76	243·49
	B	15·83	11·05	6·07	1·89	19·01	174·92	96·09	29·92	300·93
	C	13·17	9·77	5·19	1·98	16·94	128·67	68·35	26·08	223·10
	D	15·42	8·93	4·37	1·91	15·21	137·70	67·39	29·45	234·54
	E	14·00	8·51	5·22	1·68	15·41	119·14	73·08	23·52	215·74
	F	10·50	9·17	4·71	1·99	15·87	96·29	49·46	20·90	166·64
	G	10·67	9·13	4·70	2·04	15·87	97·42	50·15	21·77	169·33
	H	10·42	11·44	4·94	2·23	18·61	119·20	51·47	23·24	193·92
	I	9·33	8·68	4·96	2·00	15·64	80·98	46·28	18·66	145·92
	J	12·25	10·40	3·58	1·62	15·60	127·40	43·86	19·85	191·10
	K	11·00	10·05	4·85	1·91	16·81	110·55	53·35	21·01	184·91
	L	11·58	8·60	5·09	1·77	15·46	99·59	58·94	20·50	179·03
	M	9·67	10·48	5·08	1·68	17·24	101·34	49·12	16·25	166·71
	N	11·75	11·04	5·44	1·50	17·98	129·71	63·92	17·63	211·26
Totals and averages	14	168·59	9·93	5·01	1·83	16·77	1,673·58	844·52	308·54	2,866·63

Results obtained from 127 moiled samples of Garnet Rock from 12 trenches: Garnet Deposit, Depot Harbour—Continued.

Trench No.	Sample No.	Width of sample, feet	Percentage garnet recovered				Percentage × width			
			−20+40	−40+100	−100	Total	−20+40	−40+100	−100	Total
10.....	A	7·00	8·16	3·91	1·83	13·90	57·11	27·37	12·81	97·30
	B	11·00	7·18	3·54	1·39	12·11	78·98	38·94	15·29	133·21
	C	9·50	6·67	3·16	1·35	11·18	63·37	30·02	12·83	106·21
	D	12·00	5·38	3·82	1·13	10·33	64·56	45·84	13·56	123·96
	E	11·83	6·78	3·67	1·12	11·57	80·21	43·42	13·25	136·87
	F	10·00	6·46	4·24	1·55	12·25	64·60	42·40	15·50	122·50
	G	10·33	7·75	4·18	1·53	13·46	80·06	43·18	15·80	139·04
	H	9·33	6·68	3·88	1·60	12·16	62·32	36·20	14·93	113·45
	I	11·00	4·87	5·16	1·88	11·91	53·57	56·76	20·68	132·11
	J	10·00	5·54	4·24	1·52	11·30	55·40	42·40	15·20	113·00
	K	12·67	9·42	4·92	1·58	15·92	119·35	62·34	20·02	201·70
	L	14·42	7·15	3·74	1·27	12·16	102·89	53·93	18·31	175·35
	M	12·00	8·40	4·56	1·58	14·54	100·80	54·72	18·96	174·48
	N	13·58	7·70	4·68	2·23	14·61	104·57	63·55	30·28	198·40
	O	14·00	5·88	3·69	1·55	11·12	82·32	51·66	21·70	155·68
	P	14·00	7·35	3·26	1·11	11·72	102·90	45·63	15·54	164·08
	Q	16·75	7·97	4·43	1·55	13·95	133·50	74·20	25·96	233·66
Totals and averages	17	199·41	7·05	4·07	1·51	12·64	1,406·51	812·56	301·94	2,521·01
11.....	A	10·17	8·15	5·05	1·70	14·90	82·89	51·35	17·28	151·53
	B	13·00	9·25	4·55	1·94	15·74	120·25	59·15	25·22	204·62
	C	14·00	8·33	4·28	1·94	14·55	116·62	59·92	27·16	203·70
	D	13·42	8·44	5·19	2·23	15·86	113·26	69·65	29·93	212·84
	E	14·00	9·70	6·26	2·13	18·09	135·80	87·64	29·82	253·26
	F	15·00	10·13	6·75	3·15	20·03	151·95	101·25	47·25	300·45
	G	15·50	8·21	5·88	2·20	16·29	127·26	91·14	34·10	252·50
	H	13·83	8·00	4·45	2·44	14·89	110·64	61·54	33·75	205·93
	I	13·42	8·06	4·24	1·86	14·16	108·17	56·90	24·96	190·03
	J	13·25	7·08	4·78	2·17	14·03	93·81	63·34	28·75	185·90
	K	13·00	9·12	4·36	1·62	15·10	118·56	56·68	21·06	196·30
	L	12·67	8·68	4·29	1·46	14·43	109·98	54·35	18·50	182·83
	M	13·00	7·69	4·38	1·82	13·89	99·97	56·94	23·66	180·57
	N	12·75	8·53	4·67	1·69	14·89	108·76	59·54	21·55	189·85
	O	15·58	7·33	3·15	1·49	11·97	114·20	49·08	23·21	186·49
Totals and averages	15	202·59	8·42	4·83	2·00	15·25	1,712·12	978·47	406·20	3,096·80
12.....	A	14·00	9·68	4·35	1·49	15·52	135·52	60·90	20·86	217·28
	B	12·00	10·41	4·92	1·69	17·02	124·92	59·04	20·28	204·24
	C	9·00	5·44	4·31	2·25	12·00	48·96	38·79	20·25	108·00
	D	10·00	7·34	3·55	1·08	11·97	73·40	35·50	10·80	119·70
	E	12·00	7·46	4·00	1·48	12·94	89·52	48·00	17·76	155·28
	F	11·58	6·88	3·66	1·83	12·37	79·67	42·38	21·19	143·24
	G	13·00	10·01	4·13	2·24	16·38	130·13	53·59	29·12	212·94
Totals and averages	7	81·58	8·36	4·15	1·72	14·23	682·12	338·30	140·2	61,160·68

Magnetic separation tests on laboratory concentration products.—Three samples of the garnet concentrates from the small tests, representing the three sizes of concentrates produced, were obtained by cutting each sample into two portions, one of which was returned to the sample envelope from which it was taken, and the other used to make up the sample for the magnetic separation tests. In this manner a sample of 32 pounds representing −20+40 concentrate; a sample of 16 pounds representing −40 +100 concentrate; and a sample of 6·5 pounds representing the −100 concentrate was obtained. Each of these sizes was run through a magnetic separator, making a finished garnet product and a tailing. The results are given in the following table:—

Material	Weight		Magnetic product		Non-magnetic product		Per cent by weight final concentrate
	Pounds	Per cent	Pounds	Per cent	Pounds	Per cent	
−20 +40	32·0	58·7	26·0	81·3	6·0	18·7	59·1
−40 +100	16·0	29·4	13·0	81·3	3·0	18·7	29·5
−100	6·5	11·9	5·0	76·9	1·5	23·1	11·4
Totals and averages	54·5	100·0	44·0	80·7	10·5	19·3	100·0

This test showed that the concentrate produced on the small table still contained some gangue, and that this gangue could be eliminated by the use of magnetic separators, removing the garnet from the gangue. The gangue material remaining in the table concentrate consists of material approaching the specific gravity of garnet. In the −20+40 concentrate, 81·3 per cent is clean garnet concentrate; in the −40+100 concentrate, also 81·3 per cent; and in the −100 concentrate, 76·9 per cent. The percentage of garnet recovered as recorded in the small table tests, will have to be corrected to give the actual recovery as finished concentrate. From the above table it will be seen that the finished concentrate consists of 59·1 per cent −20+40 mesh, 29·5 per cent −40+100 mesh, 11·4 per cent −100 mesh. It is also seen that table concentration gives as clean a product on the −40+100 material as on the −20+40 material.

The following table gives the actual recovery of garnet in finished concentrate, as determined by the magnetic concentration tests on the table concentrate. In each case the recoveries indicated in former tables are multiplied by 81·3 per cent in the −20+40; by 81·3 per cent in the −40+100; and by 76·9 per cent in the −100. This is done for the twelve trenches.

Table giving Correction Results of Small Scale Work

BLOCK "A"

Trench No.	Number of samples	Width of sample, feet	Percentage—finished concentrate			
			−20+40 factor 81·3	−40 +100 factor 81·3	−100 factor 76·9	Total factor 80·7
1	12	100·25	5·93	3·52	1·35	10·79
2	12	152·73	6·80	3·65	1·25	11·69
3	5	62·58	6·65	3·46	1·24	11·31
4	14	184·24	6·37	3·31	1·18	10·84
5	8	79·33	6·73	3·35	1·25	11·31
6	8	125·16	8·60	3·70	1·24	13·51
Totals and averages	59	704·29	6·83	3·49	1·25	11·57

BLOCK "B"

7	8	105·51	7·89	3·87	1·49	13·24
8	7	100·92	7·33	3·66	1·41	12·39
9	14	168·59	8·07	4·07	1·41	13·53
Totals and averages	29	375·02	7·76	3·87	1·43	13·06

BLOCK "C"

10	17	199·41	5·73	3·31	1·16	10·20
11	15	202·59	6·85	3·93	1·54	12·31
12	7	81·58	6·80	3·37	1·32	11·48
Totals and averages	39	483·58	6·45	3·54	1·34	11·33

BLOCKS "A", "B" AND "C"

1–6	59	704·29	6·84	3·50	1·25	11·57
7–9	29	375·02	7·76	3·87	1·44	13·06
10–12	39	483·58	6·46	3·54	1·34	11·33
Totals and averages	127	1,562·89	6·97	3·61	1·31	11·89

Percentages obtained from Weights of Samples

Trench No.	Number of samples	Weight of samples, less small samples	Percentage garnet recovered				Weight percentage of garnet recovered
			Total	−20+40	−40+100	−100	
1	12	1,368	13·27	7·23	4·30	1·74	18,249·95
2	12	1,132	14·47	8·37	4·48	1·62	16,383·72
3	5	511	14·65	8·52	4·44	1·69	7,486·78
4	14	1,313	13·44	7·83	4·06	1·54	17,642·54
5	8	691	14·04	8·29	4·13	1·62	9,703·04
6	8	695	16·96	10·72	4·61	1·63	11,784·81
7	8	777	16·53	9·78	4·50	1·95	12,847·34
8	7	683	15·05	8·83	4·41	1·81	10,277·52
9	14	1,415	16·67	9·87	4·98	1·82	23,594·67
10	17	1,466	12·55	7·00	4·03	1·51	18,405·18
11	15	1,311	15·28	8·44	4·84	2·00	20,026·06
12	7	617	13·96	8·20	4·07	1·69	8,611·32
Totals and averages	127	11,979	14·74	8·59	4·41	1·71	174,994·93
			Factor 80·7	Factor 81·3	Factor 81·3	Factor 76·9	
Correction for finished concentrate			11·89	6·98	3·59	1·32	

Procedure followed for determining methods of treatment.—After each of the samples had been crushed to one-half inch and the small sample cut out for the small scale laboratory tests to determine the garnet content, the remaining portion of the samples from each trench were mixed together, making twelve large samples, each representing a trench. These lots were used for the larger tests to determine methods of treatment, and as the tests made were not all similar in methods of procedure, each test is described separately below in the order in which they were run.

Trench No. 10.—After the reduction of the samples to 12 mesh, some mica was removed by a light suction. One-half of the 12 mesh material was classified into three products and a classified product fed to the concentrating table. The other half was sized and a sized product fed to the table. This was done to compare the work of the table on a classified and sized feed.

Trench No. 11.—After reduction to 12 mesh and the removal of some mica by a light suction, the material was screened wet on 34 mesh, and 80 mesh Callow belt screens, and each size tabled.

Trench No. 9.—Procedure similar to Trench No. 11.

Trench No. 4.—After reduction to 12 mesh, the mica was removed by medium suction. The −12 mesh material was sized wet on Callow belt screens, 34 and 80 mesh, and each size run separately over a Wilfley concentrator. A sample of the mica product was screened and each size tabled to determine the garnet content in the mica product.

Trench No. 2.—After reduction to 12 mesh, the mica was removed by a strong suction. As very little fines remained in the −12 mesh material, it was screened wet on a 34 mesh Callow belt screen only, and the two

sizes made were run separately over a Wilfley concentrator. A sample of the mica product was sized and each size tabled to determine the garnet content of this product.

Trench No. 1.—Same procedure followed as on the samples from Trench No. 2. Suction used was not quite as strong.

Trench No. 5.—Same procedure followed as on the samples of Trench No. 1.

Trench No. 6.—Same procedure as on the samples of Trench No. 1.

Trench No. 7.—Same procedure as for samples of Trench No. 1.

Trench No. 8.—Same procedure as for samples of Trench No. 1.

Trench No. 12.—After reduction to 12 mesh and the removal of a mica product by suction, the material was classified to remove any slime, sized wet on a 34 mesh Callow belt screen, and each size run separately over a Wilfley concentrator, making a concentrate, middling, and tailing. In the case of the coarser size, the concentrate was screened on a 24 mesh Callow belt screen, making a finished concentrate and a middling. The middling from the first run over the table was re-run, making a concentrate and tailing. This concentrate was screened on a 24 mesh Callow belt screen, making a finished concentrate and middling. The middlings, or throughs, from the screening were added together and re-tabled, making a concentrate and tailing, the concentrate from which was screened on a 34 mesh Callow belt screen, making a finished concentrate and middling. The middling from this operation was added to the −34 mesh material to be tabled. A similar procedure was followed with the −34 mesh size, except that screening was done on a 44 mesh Callow belt screen.

Trench No. 3.—Same procedure followed as for samples of Trench No. 12, with the exception that no mica product was removed.

Procedure followed and results of test work on samples from Trench No. 10.—The samples from this trench were crushed in rolls set at $\frac{1}{8}''$, and screened on a Ferraris shaking screen fitted with 12 mesh screen, and with suction take off at the discharge end, for the removal of the mica. The oversize was returned to the rolls, until all passed through the screen or was removed by the suction. The suction used was low so that only a small amount of mica was removed in this manner.

Product	Weight pounds	Per cent of feed
Suction	93	6·46
−12 mesh	1,288	89·44
Dust loss	59	4·10
Feed	1,440	100·00

The −12 mesh material was cut into two equal portions. One was classified into three products in a two-spigot launder classifier, and each run separately over the Wilfley concentrator, giving the following results:—

Product	Weight pounds	Per cent of heads
Coarse concentrate	45·5	6·32
Medium concentrate	44·5	6·18
Fine concentrate	6·5	0·90
Total concentrates	96·5	13·40

The other portion was screened wet on 34 mesh and 80 mesh Callow belt screens, and each size fed separately to the table, giving the following results:—

Product	Weight pounds	Per cent of feed
−12+34 concentrate	62·0	8·61
−34+80 concentrate	24·0	3·33
−80 concentrate	11·5	1·60
Total concentrates	97·5	13·54

A comparison of the concentrates produced in the above tests on two equal portions of the −12 mesh material shows that the larger amount and better grade of concentrate was obtained on a sized feed. It was found that the table worked better on a sized feed than on a classified feed, so that a sized feed was used in all subsequent tests.

Procedure followed and results of test work on samples from Trench No. 11.—The samples from this trench were reduced to 12 mesh and mica removed as in the preceding tests, with the exception that very low suction was used, taking off less mica product.

Product	Weight pounds	Per cent of feed
Suction	11·5	0·88
−12 mesh	1,226·5	93·55
Dust loss	73·0	5·57
Feed	1,311·0	100·00

The 12 mesh material was screened wet on 34 mesh and 80 mesh Callow belt screens, and each size fed separately to the table, with the following results:—

Product	Weight pounds	Per cent of heads
−12+34 concentrate	128·25	9·78
−34+80 concentrate	57·00	4·35
−80 concentrate	24·75	1·89
Total concentrates	210·00	16·02

Procedure followed and results of test work on samples of Trench No. 9.—The same procedure was followed as with the samples from Trench No. 11. The suction used was similar to that employed on the samples from Trench No. 10.

Product	Weight pounds	Per cent of heads
Suction	80·00	5·66
−12 mesh	1,276·75	90·29
Dust loss	57·25	4·05
Feed	1,414·00	100·00

The 12 mesh material was screened wet on 34 mesh and 80 mesh Callow belt screens, and each size tabled separately:—

Product	Weight pounds	Per cent of heads
−12+34 concentrate	155·5	11·00
−34+80 concentrate	58·5	4·14
−80 concentrate	25·5	1·80
Total concentrates	239·5	16·94

Procedure followed and results of test work on samples of Trench No. 4.—The samples from this trench were reduced to 12 mesh in a similar manner to the preceding ones, with the exception that a medium suction was used at the feed end of the screen, and some oversize, consisting of mica, was not fed back to the rolls.

Product	Weight pounds	Per cent of feed
Suction	213·0	16·22
−12 mesh	1,034·0	78·75
+12 mesh (mica)	7·5	0·57
Dust loss	58·5	4·46
Feed	1,313·0	100·00

The 12 mesh material was sized wet on 34 mesh and 80 mesh Callow belt screens and each size tabled:—

Product	Weight pounds	Per cent of heads
−12+34 concentrate	117·00	8·91
−34+80 concentrate	37·25	2·84
−80 concentrate	12·50	0·95
Total concentrates	166·75	12·70

As the suction was applied at the feed end of the shaking screen, a certain amount of fine garnet was drawn up with the mica into the collecting tank. A sample of the suction product was screened, and the various sizes tabled to determine the percentage of garnet in this mica product. It was found to contain 0·68 per cent of the weight of the sample of Trench No. 4. This added to the 12·70 per cent recovered in the table concentrate would give as value for the trench, 13·38 per cent garnet concentrate. The suction product contained 4·21 per cent garnet, 0·09 per cent coarser than 35 mesh, 4·12 finer than 35 mesh. Any garnet lost in the suction product was very fine, the greater proportion of which would be lost in subsequent tabling operations if left in the material fed to the table.

Procedure followed and results of test work on samples from Trench No. 2.—The samples from this trench were crushed to 12 mesh in a similar manner to those of Trench No. 10. A strong suction was used to remove the mica at the feed end of the shaking screen:—

Product	Weight pounds	Per cent of feed
Suction	563·5	49·15
−12 mesh	519·0	45·27
Dust loss	64·0	5·58
Feed	1,146·5	100·00

The strong suction used removed almost 50 per cent of the feed to the screen, and delivered to the collector the mica and fines. It was, therefore, not necessary to make three sized products for the table. The −12 mesh material was only sized wet on a 35 mesh Callow belt screen.

Product	Weight pounds	Per cent of heads
−12+34 concentrate	101·75	9·02
−34 concentrate	21·25	1·88
Total concentrates	123·00	10·90

A small scale test was made on a sample of the suction product. It was found to contain garnet representing 3·70 per cent of the weight of the samples of Trench No. 2. This added to the 10·90 per cent recovered in table concentrate would give a value for the trench of 14·60 per cent garnet concentrate. The suction product contained 7·53 per cent garnet, 1·94 per cent +35 mesh, and 3·59 per cent −35 mesh. These results show that too strong a suction was used, removing too much garnet with the mica.

Procedure followed and results of test work on samples from Trench No. 1. —The samples from this trench were crushed to 12 mesh in a similar manner to those of Trench No. 10. A strong suction was used to remove the mica at the feed end of the shaking screen. The suction was not quite as strong as that used in the preceding test on the samples of Trench No. 2.

Product	Weight pounds	Per cent of feed
Suction	609·0	44·57
−12 mesh	687·0	50·27
Dust loss	70·5	5·16
Feed	1,366·5	100·00

The −12 mesh material was sampled and the balance, 681·5 pounds, was screened wet on a 34 mesh Callow belt screen, and the two sizes fed separately to a Wilfley concentrator.

Product	Weight pounds	Per cent of heads
−12+34 concentrate	116·0	8·55
−34 concentrate	31·0	2·29
Total concentrates	147·0	10·84

A small scale test was made on a sample of the suction product. It was found to contain garnet representing 2·25 per cent of the weight of the samples of Trench No. 1. This percentage added to the 10·84 per cent recovered in table concentrate would give a value for the trench of 13·09 per cent garnet concentrate. The suction product contained 5·06 per cent garnet, 0·67 per cent +35 mesh, and 4·39 per cent −35 mesh.

Procedure followed and results of test work on samples from Trench No. 5. —Same procedure followed as on samples from Trench No. 1.

Screen product	Weight pounds	Per cent of heads
Suction	292·0	42·35
−12 mesh	349·0	50·62
Dust loss	48·5	7·03
Feed	689·5	100·00

Table product	Weight pounds	Per cent of heads
−12+34 concentrate	60·0	8·85
−34 concentrate	14·5	2·14
Total concentrates	74·5	10·99

Suction product..... 0·75% +35 mesh
5·14% −35 mesh

Total..... 5·89% garnet,
representing 2·49 per cent of heads, making a total value for trench −13·48 per cent.

Procedure followed and results of test work on samples from Trench No. 6. —Same procedure followed as on the samples from Trench No. 1.

Screen product	Weight pounds	Per cent of heads
Suction	296·0	42·68
−12 mesh	358·5	41·70
Dust loss	39·0	5·62
Feed	693·5	100·00

Table product	Weight pounds	Per cent of heads
−12+34 concentrate	81·00	11·88
−34 concentrate	14·75	2·16
Total concentrates	95·75	14·04

Suction product..... 1·55% +35 mesh
5·49% −35 mesh

Total..... 7·04% garnet,
representing 3·00 per cent of heads, making a total value for trench, 17·04 per cent.

Procedure followed and results of test work on samples from Trench No. 7. —Same procedure followed as on the samples from Trench No. 1.

Screen product	Weight pounds	Per cent of heads
Suction	381·0	48·78
−12 mesh	378·0	48·40
Dust loss	22·0	2·82
Feed	781·0	100·00

Table product	Weight pounds	Per cent of heads
−12+34 concentrate	87·5	11·37
−34 concentrate	18·5	2·40
Total concentrates	106·0	13·77

Suction product..... 1·25% +35 mesh
5·66% −35 mesh

Total..... 6·91% garnet,
representing 3·37 per cent of heads, making a total value for trench, 17·14 per cent.

Procedure followed and results of test work on samples from Trench No. 8. —Same procedure as on the samples from Trench No. 1.

Screen product	Weight pounds	Per cent of heads
Suction	311·0	45·17
−12 mesh	357·0	51·85
Dust loss	20·5	2·98
Feed	688·5	100·00

Table product	Weight pounds	Per cent of heads
−12+34 concentrate	68·50	10·10
−34 concentrate	17·75	2·62
Total concentrates	86·25	12·72

Suction product..... 0·98% +35 mesh
6·20% −35 mesh

Total..... 7·18% garnet,
representing 3·24 per cent of heads, making a total value for trench, 15·96 per cent

Procedure followed and results of test work on samples from Trench No. 12 —The samples from this trench were crushed in rolls set at $\frac{1}{8}$" and screened on a Ferraris shaking screen fitted with a 12 mesh screen and with suction take off at the feed end. The oversize was returned to the rolls until all passed through the screen.

Product	Weight pounds	Per cent of feed
Suction	299·5	48·50
−12 mesh	301·0	48·75
Dust loss	17·0	2·75
Feed	617·5	100·00

The −12 mesh material was fed to a Dorr drag classifier to remove any slimes, and the sand screened on a 34 mesh Callow belt screen, making two sizes, which were fed separately to a Wilfley concentrator. The subsequent procedure can best be illustrated by the following flow sheet:—

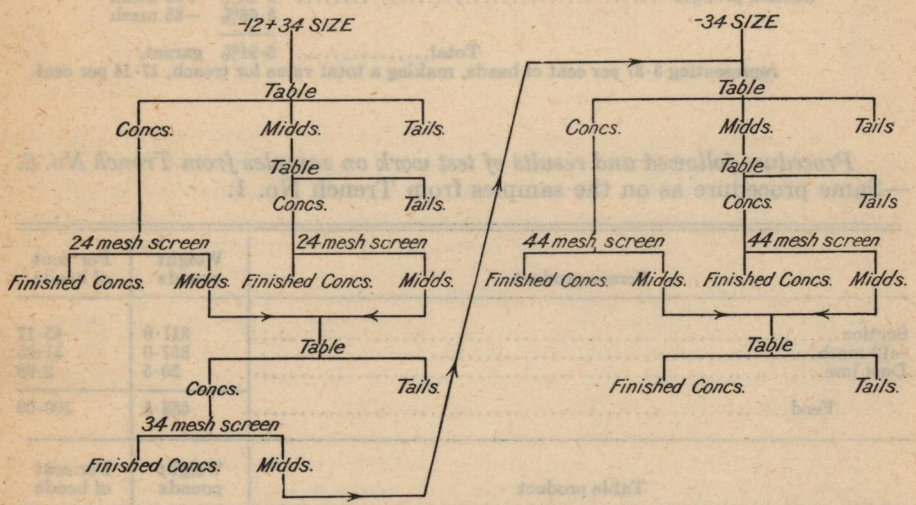


FIG. 8. Flow sheet showing procedure in testing samples of garnet rock from Trench No. 12, Depot Harbor, Ont.

294·5 pounds of −12 mesh material was fed to the classifier. There were 8·5 pounds of slime overflow, and 286 pounds of sands for concentration. 1½ pound of material was cleaned up after the test.

Product	Weight pounds	Per cent of heads
+24 concentrate	19·5	3·23
+24 concentrate from middling	15·5	2·57
+34 concentrate	7·5	1·24
+44 concentrate	2·5	0·41
+44 concentrate from middling	4·0	0·66
−44 concentrate	4·5	0·75
Total concentrates	53·5	8·86

Suction product	0·71%	+35 mesh
	5·84%	−35 mesh
Total	6·55%	garnet,

representing 3·18 per cent of heads, making a total value for trench, 12·04 per cent

Procedure followed and results of test work on samples from Trench No. 3.—The samples from this trench were crushed in rolls set at $\frac{1}{8}''$ and screened on a Ferraris shaking screen fitted with 12 mesh screen. No suction was used to remove the mica. The oversize was returned to the rolls until all passed through the screen.

Product	Weight pounds	Per cent of heads
−12 mesh	483·0	92·26
Dust loss	40·5	7·74
Heads	523·5	100·00

The −12 mesh material was fed to a Dorr drag classifier to remove the slimes, and the sands screened on a 34 mesh Callow belt screen, making two sizes, which were fed separately to a Wilfley concentrator. The subsequent procedure is the same as that given in the flow sheet under trench No. 12. 474·5 pounds of −12 mesh material was fed to the classifier. There were 15 pounds of slime overflow, and 459·5 pounds of sands for concentration. 0·5 pound of material was cleaned up after the test.

Product	Weight pounds	Per cent of heads
+24 concentrate	17·5	3·41
+24 concentrate from middling	12·5	2·43
+34 concentrate	7·5	1·46
+44 concentrate	7·0	1·36
+44 concentrate from middling	3·0	0·58
−44 concentrate	16·0	3·11
Total concentrates	63·5	12·35

This test and the preceding one were made to determine whether a finished concentrate could not be produced by screening the table concentrate and re-tabling the throughs. It was found that the screened product was very clean, but that it was difficult to obtain a clean concentrate by re-tabling the throughs or middlings, as the gangue material contained in them was of almost the same specific gravity as the garnet. These points were determined by magnetic separation tests made on the concentration products.

SUMMARY OF RESULTS FROM LARGE SCALE TESTS ON TRENCH SAMPLES

Trench No.	Per cent concentrates				
	−12+34	−34+80	−80	In suction product	Total
10	8·61	3·33	1·60	.....	13·54
11	9·78	4·35	1·89	.....	16·02
9	11·00	4·14	1·80	.....	16·94
4	8·91	2·84	0·95	0·68	13·38
2	9·02	1·88	.....	3·70	14·60
1	8·55	2·29	.....	2·25	13·09
5	8·85	2·14	.....	2·49	13·48
6	11·88	2·16	.....	3·00	17·04
7	11·37	2·40	.....	3·37	17·14
8	10·10	2·62	.....	3·24	15·96
12	7·04	1·82	.....	3·18	12·04
3	7·30	5·05	.....	.....	12·35

NOTE.—The percentages given under the column −34+80 for Trench Nos. 2, 1, 5, 6, 8, 12, and 3, are for −34 mesh concentrates.

On examination of the figures for total percentages, it will be found that these correspond very closely with the results obtained from the small scale tests, showing that the small laboratory results are practically the same as that obtained from actual milling operations.

Magnetic separation tests on concentration products.—These tests were performed to eliminate the non-magnetic gangue remaining in the table concentrate, and to determine the percentage of finished concentrate in the table products. As shown in preceding tests, the table concentrate contained some gangue which was close to the specific gravity of the garnet and reported on the table with the garnet in the concentrate.

Test No. 1.—This test was run on three samples representing the +34, −34+80, and −80 concentrates from all the trench samples, except those of Trenches Nos. 12 and 3. The samples were run dry through an Ullrich magnetic separator.

Product	+34		−34+80		−80	
	Weight		Weight		Weight	
	lbs.	%	lbs.	%	lbs.	%
Concentrate	70·25	81·2	70·50	82·0	15·50	77·5
Tailing	16·25	18·8	15·50	18·0	4·50	22·5
Feed	86·50	100·0	86·00	100·0	20·00	100·0

It will be noted that these results correspond very closely to those obtained from the concentrate of the small scale tests. For comparison the two results are again given:—

Small scale tests		Large scale tests	
−20+40	81·3 per cent F.C.	−12+34	81·2 per cent F.C.
−40+100	81·3 per cent F.C.	−34+80	82·0 per cent F.C.
−100	76·9 per cent F.C.	−80	77·5 per cent F.C.

Test No. 2.—This test was run on the products obtained by screening the +34 mesh concentrate on 28 mesh to determine whether a finished concentrate could be obtained by screening.

Product	+28		−28	
	Weight		Weight	
	lbs.	%	lbs.	%
Concentrate	65·5	99·3	9·5	50·0
Tailing	0·5	0·7	9·5	50·0
Feed	66·0	100·0	19·0	100·0

The results of this test show that a clean finished product is made by screening the 34 mesh table concentrates on 28 mesh, that practically all the gangue passes through the screen and this product is 50 per cent gangue, 50 per cent garnet. The −28 mesh material would require re-cleaning by the use of a magnetic separator.

Test No. 3.—This test was run on the products obtained by screening the −34+80 concentrates on 48 mesh to determine whether a finished concentrate could be obtained by screening.

Product	+48		−48	
	Weight		Weight	
	lbs.	%	lbs.	%
Concentrate	49·5	95·2	20·5	62·1
Tailing	2·5	4·8	12·5	37·9
Feed	52·0	100·0	33·0	100·0

The results of this test show that a clean finished product is made by screening the −34+80 table concentrates on 48 mesh; that practically all the gangue passes through the screen, and this product is 37·9 per cent gangue, 62·1 per cent garnet. The −48 mesh material would require re-cleaning by the use of a magnetic separator.

Test No. 4.—This test was run on the products obtained by screening the −80 table concentrates on 100 mesh to determine whether a finished concentrate could be obtained by screening.

Product	+100		−100	
	Weight		Weight	
	lbs.	%	lbs.	%
Concentrate	8·5	77·3	6·5	68·4
Tailing	2·5	22·7	3·0	31·6
Feed	11·0	100·0	9·5	100·0

The results of this test show that a finished product cannot be made by screening the −80 mesh table concentrate on 100 mesh. The +100 mesh contains 22·7 per cent gangue, and the −100 31·6 per cent gangue. The −80 table concentrate would require re-cleaning by the use of a magnetic separator.

Test No. 5.—A test was run on samples of the table concentrates +34, −34+80 and −80 on the Ullrich magnetic separator, using a wet feed instead of a dry one. A stronger magnetic field was used. The results obtained were practically the same as those from the dry feed, possibly a slightly cleaner concentrate was made, but the tailing seemed to contain a little more garnet. If the magnetic field had been slightly stronger, the results would have been very similar.

The Possibilities of Recovering Mica Products Suitable for the Trade.—In the majority of the tests made on the trench samples, a mica product was obtained by suction, similarly to the practice for the removal of asbestos fibre from the crushed rock. In some cases this mica product represented as high as 50 per cent of the feed when a strong suction was used, but contained in such cases a considerable amount of fine garnet. However, as the saleable mica product would be sizes coarser than 35 mesh, the fines could be returned to the classifier with the rest of the feed and the fine garnet saved. When a moderate suction was employed very little garnet entered this product. The possibility of recovering a mica product would depend on the rock being dry. When the rock contained much moisture this system could not be used. However, it would be well to make provision for the recovery of a mica product in the construction of a mill, as during certain seasons of the year the rock would be in a condition favourable for its recovery, and certain grades could be produced, with a possibility of their use for certain purposes. Samples of the suction product from several of the tests were sized, and the sizes, coarser than 35 mesh, gave a clean product that might be used profitably, and might constitute a valuable by-product.

From the results of the above tests for the determination of methods of treatment for the recovery of the garnet from the rock, in products suitable for the trade, the following flow sheet, Fig. 9, is suggested:—

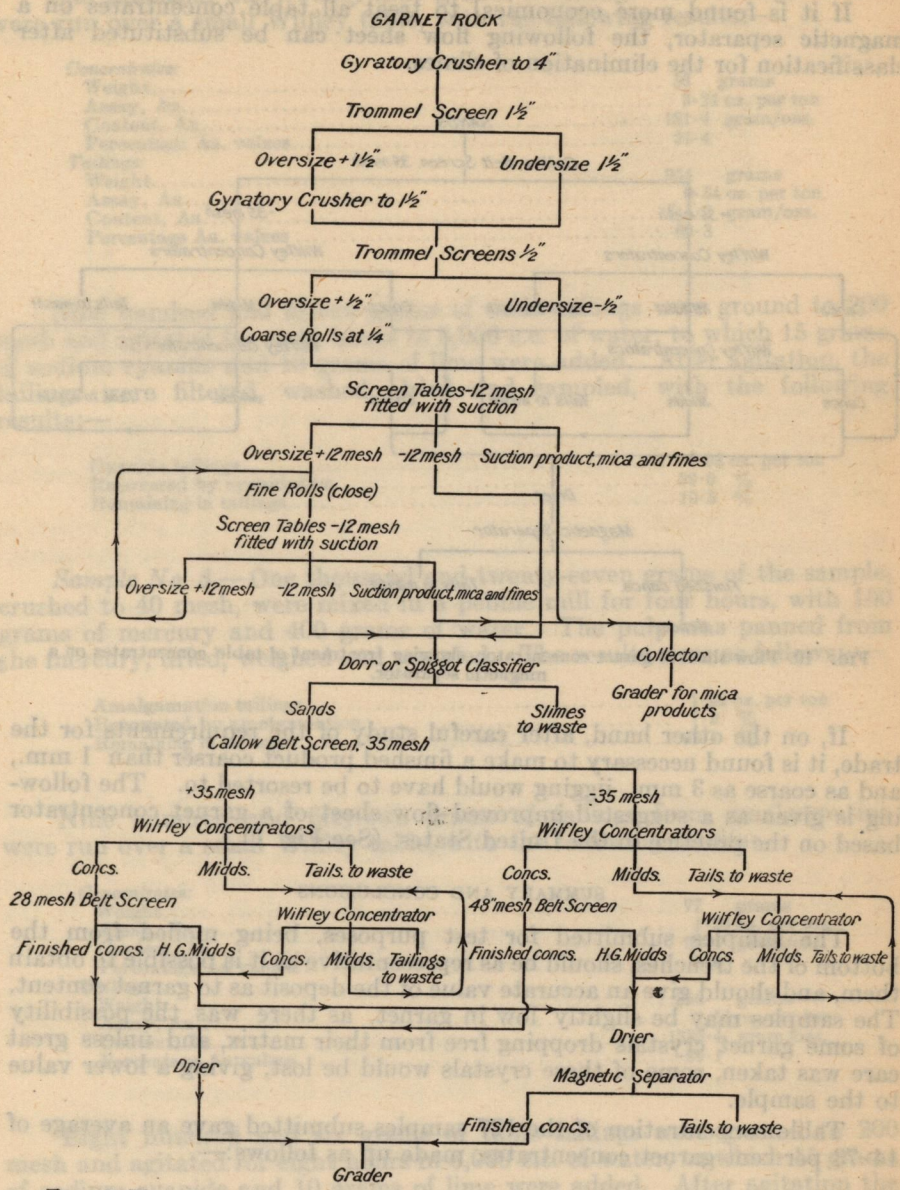


FIG. 9. Flow sheet of garnet concentrator, showing methods of treatment for recovery of garnet from the rock.

If it is found more economical to treat all table concentrates on a magnetic separator, the following flow sheet can be substituted after classification for the elimination of slimes.

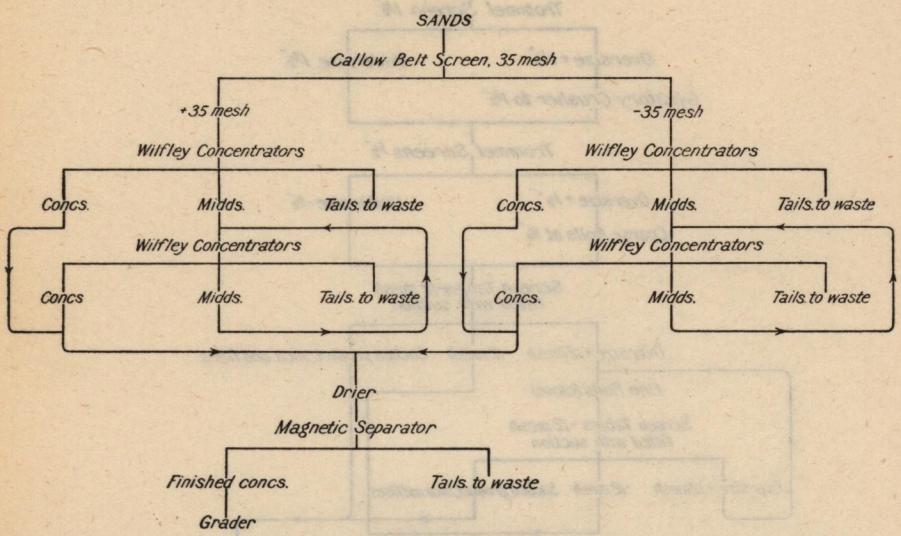


FIG. 10. Flow sheet of garnet concentrator, showing treatment of table concentrates on a magnetic separator.

If, on the other hand, after careful study of the requirements for the trade, it is found necessary to make a finished product coarser than 1 mm., and as coarse as 3 mm., jigging would have to be resorted to. The following is given as a suggested improved flow sheet of a garnet concentrator based on the practice in the United States. (See Fig. 11.)

SUMMARY AND CONCLUSIONS

The samples submitted for test purposes, being moiled from the bottom of the trenches, should be as representative as it is possible to obtain them, and should give an accurate value of the deposit as to garnet content. The samples may be slightly low in garnet, as there was the possibility of some garnet crystals dropping free from their matrix, and unless great care was taken, some of these crystals would be lost, giving a lower value to the sample.

Table concentration of the 127 samples submitted gave an average of 14·73 per cent garnet concentrates, made up as follows:—

	%
−20+ 40 concentrate	8·58
−40+100 "	4·44
−100 "	1·71

The results of the large scale tests on the trench samples gave practically the same results.

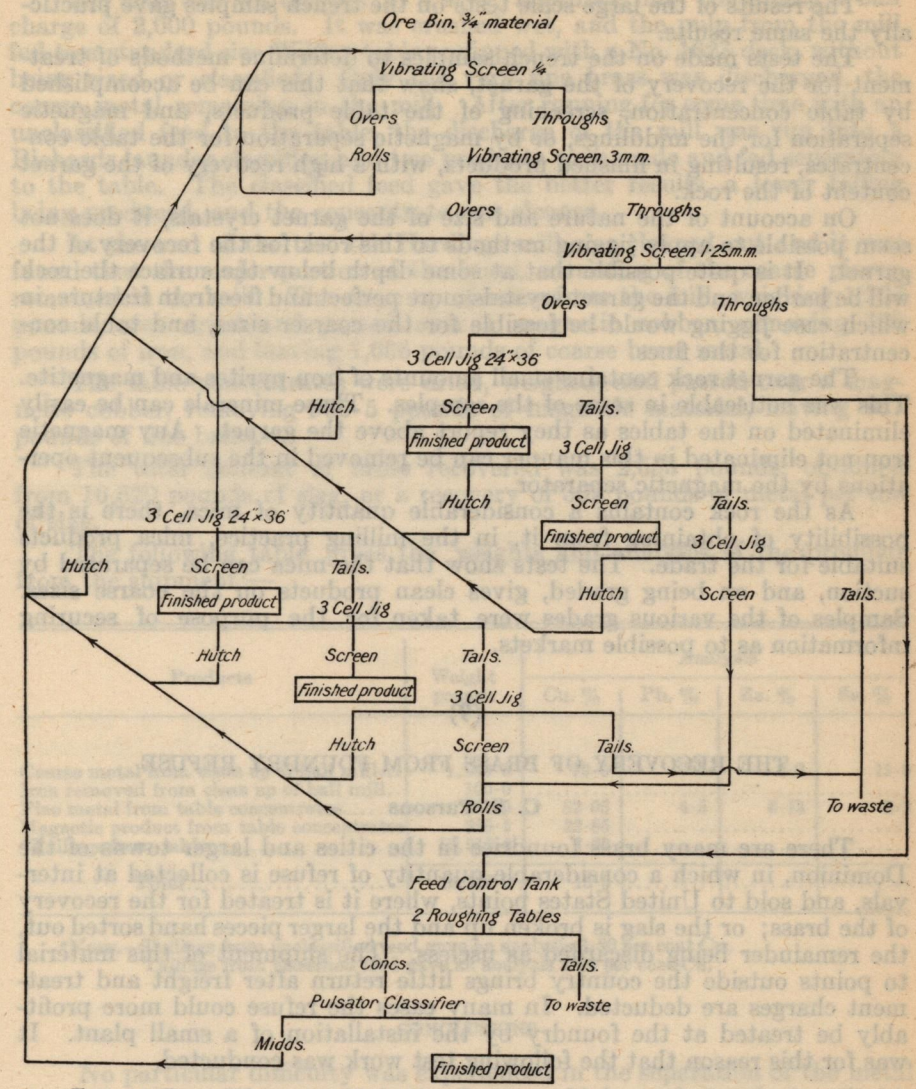


FIG. 11. Suggested improved flow sheet of garnet concentrator based on practice in the United States.

The table concentrates contain some gangue, difficult to remove by table concentration, on account of its specific gravity being very close to that of the garnet. This can be eliminated by screening, and by the use of the magnetic separator, giving a finished product. The finished concentrates show an average of 11·89 per cent, made up as follows:—

	%
−20+ 40 concentrate	6·97
−40+100 "	3·61
−100 "	1·31

The results of the large scale tests on the trench samples gave practically the same results.

The tests made on the trench samples to determine methods of treatment for the recovery of the garnet, show that this can be accomplished by table concentration, screening of the table products, and magnetic separation for the middlings, or by magnetic separation for the table concentrates, resulting in finished products, with a high recovery of the garnet content of the rock.

On account of the nature and size of the garnet crystals, it does not seem possible to apply jigging methods to this rock for the recovery of the garnet. It is quite possible that at some depth below the surface the rock will be harder, and the garnet crystals more perfect and free from fracture, in which case jigging would be feasible for the coarser sizes, and table concentration for the fines.

The garnet rock contains small amounts of iron pyrites and magnetite. This was noticeable in some of the samples. These minerals can be easily eliminated on the tables as they report above the garnet. Any magnetic iron not eliminated in this manner can be removed in the subsequent operations by the magnetic separator.

As the rock contains a considerable quantity of mica, there is the possibility of obtaining from it, in the milling practice, mica products suitable for the trade. The tests show that the mica can be separated by suction, and on being graded, gives clean products on the coarse sizes. Samples of the various grades were taken for the purpose of securing information as to possible markets.

(9)

THE RECOVERY OF BRASS FROM FOUNDRY REFUSE

C. S. Parsons

There are many brass foundries in the cities and larger towns of the Dominion, in which a considerable quantity of refuse is collected at intervals, and sold to United States points, where it is treated for the recovery of the brass; or the slag is broken up and the larger pieces hand sorted out, the remainder being discarded as useless. The shipment of this material to points outside the country brings little return after freight and treatment charges are deducted. In many cases the refuse could more profitably be treated at the foundry by the installation of a small plant. It was for this reason that the following test work was conducted.

Test No. 151

A shipment of 16,520 pounds of foundry slag and sweepings was received at the Ore Dressing and Metallurgical Laboratories on May 4, 1921, from Mr. S. R. Francis, of the Ontario Specialties, Limited, Ottawa, Ont.

Test work was desired on this shipment to obtain a separation of the metal from the slag in as coarse a form as possible, and to determine a simple flow sheet for the recovery of the metal from the slag and sweepings.

The slag was fed directly to a 4′ 6″ Hardinge mill, containing a ball charge of 2,000 pounds. It was crushed wet, and the pulp from the mill fed to a standard size Wilfley table equipped with a No. 1623 deck, without being sized or classified. Only the very fine brass was discharged, the coarse metal remaining in the mill. After running for some time with an unclassified feed to the table, the discharge of the mill was run into a Richards launder classifier, and two products were made and fed separately to the table. The classified feed gave the better results, a lower tailing being produced, and the concentrate was cleaner.

At the end of the test the Hardinge mill was cleaned out and it was found that the major portion of the brass, consisting of the coarse pieces, remained in the mill. The clean up of brass from the mill, weighing 1,766 pounds, was dried and passed over a magnetic cobber, removing 100 pounds of iron, and leaving 1,666 pounds of coarse brass metal.

The table concentrates were dried, weighed, and passed over a magnetic cobber, removing 356·5 pounds of magnetic material, leaving 887 pounds of fine brass.

The total amount of brass recovered was 2,553 pounds, obtained from 16,520 pounds of slag, or a recovery of 309 pounds of metal per ton of slag.

The following table gives the weights and analyses of the products from the shipment:—

Products	Weight pounds	Analysis			
		Cu. %	Pb. %	Zn. %	Sn. %
Coarse metal from clean up of ball mill...	1,666·0	79·0	7·8	2·0	11·0
Iron removed from clean up of ball mill..	100·0	.....	.....	.....	.....
Fine metal from table concentrates.....	887·0	82·05	4·5	6·13	6·2
Magnetic product from table concentrates	356·5	22·85	.....	.....	.....
*Tailings from table.....	13,510·5	3·20	.....	.....	.....
Total.....	16,520·0	15·5	.....	.....	.....

*Note.—Tailings from unclassified feed gave an analysis 3·80 per cent Cu.
Tailings from classified feed gave an analysis 2·70 per cent Cu.

CONCLUSIONS

No particular difficulty was experienced in the separation of the metal from the slag. The metal obtained was a high grade product, suitable for re-melting.

For the small brass foundry, a simple equipment of a small size Hardinge mill, and a half-size Wilfley table, would be sufficient to handle the slag and sweepings.

(10)

THE METALLURGICAL TREATMENT OF THE GOLD ORES OF RICE LAKE DISTRICT, MANITOBA

R. K. Carnochan

In the Summary Report of the Mines Branch for 1920, there is described under test No. 135, the experimental work conducted on a shipment of gold ore from Little Rice lake, Manitoba. That report, together with the investigation conducted on the ores from Big Rice lake given below, covers the metallurgical treatment of the gold ores from Rice Lake district, in the southeastern section of the province, as far as exploration work has proven them to date. These ores, containing copper minerals and iron sulphides, are somewhat different from the ordinary gold ores covered by metallurgical practice, and require a different treatment to recover the valuable minerals.

In the Summary Reports of the Mines Branch for 1919 and 1920, there is described under tests Nos. 127 and 132 the experimental work conducted on the gold ores from the Herb Lake district in the northwestern section of the province. The ores from this district contain chalcopyrite, arsenopyrite, pyrite, sphalerite and galena, requiring special treatment for the recovery of the valuable minerals.

Test No. 141

Two small samples of gold ore were received January 18, 1921, at the Ore Dressing and Metallurgical Laboratories, from the Pan Extension Gold Mines, Ltd., Winnipeg, Man. The samples were from the company's property at Big Rice lake, and consisted of quartz and greenstone, carrying a small amount of sulphides.

Amalgamation and concentration tests were desired to ascertain whether the ore was amenable to these methods of treatment. Cyanide tests were also conducted.

Assays of the samples gave the following:—

Sample No. 2	Au	0·64 oz. per ton
Sample No. 3	Au	1·65 oz. per ton

Sample No. 2.—One thousand one hundred and twenty-four grams of the sample, crushed to pass 40 mesh, were mixed in a pebble mill for four hours with 100 grams of mercury and 400 grams of water. The pulp was panned from the mercury, and the pulp dried, weighed and sampled. The results were as follows:—

Amalgamation tailings	0·60 oz. per ton
Recovered by amalgamation	6·3 %
Remaining in tailings	93·7 %

One thousand and twenty-nine grams of tailings from amalgamation were run over a small Wilfley table, with the following results:—

Concentrates:		
Weight	56	grams
Assay, Au	3·24	oz. per ton
Content, Au	181·4	gram/ozs.
Percentage Au. values	24·4	
Tailings:		
Weight	954	grams
Assay, Au	0·54	oz. per ton
Content, Au	515·2	gram/ozs.
Percentage Au. values	69·3	

Nine hundred and fifteen grams of table tailings were ground to 200 mesh and agitated for eight hours in 6,000 c.c. of water, to which 15 grams of sodium cyanide and 10 grams of lime were added. After agitation, the tailings were filtered, washed, dried and sampled, with the following results:—

Cyanide tailings	0·08	oz. per ton
Recovered by cyanidation	59·0	%
Remaining in tailings	10·3	%

Sample No. 3.—One thousand and twenty-seven grams of the sample, crushed to 40 mesh, were mixed in a pebble mill for four hours, with 100 grams of mercury and 400 grams of water. The pulp was panned from the mercury, dried, weighed and sampled. The results were as follows:—

Amalgamation tailings	1·52	oz. per ton
Recovered by amalgamation	7·9	%
Remaining in tailings	92·1	%

Nine hundred and fifty-two grams of tailings from amalgamation were run over a small Wilfley table, with the following results:—

Concentrates:		
Weight	77	grams
Assay, Au	1·20	oz. per ton
Content, Au	92·4	gram/ozs.
Percentage Au values	6·0	
Tailings:		
Weight	856	grams
Assay, Au	1·56	oz. per ton
Content, Au	1335·4	gram/ozs.
Percentage Au values	86·1	

Eight hundred and six grams of table tailings were ground to 200 mesh and agitated for eight hours in 6,000 c.c. of water, to which 15 grams of sodium cyanide and 10 grams of lime were added. After agitation the tailings were filtered, washed, dried and sampled, with the following results:—

Cyanide tailings	0·10	oz. per ton
Recovered by cyanidation	80·6	%
Remaining in tailings	5·5	%

SUMMARY

The following tables show the percentages of the gold values recovered:—

Sample No. 2:	
	p.c.
By amalgamation	6·3
By concentration	24·4
By cyanidation	59·0
Remaining in tailings	10·3
Sample No. 3:	
	p.c.
By amalgamation	7·9
By concentration	6·0
By cyanidation	80·6
Remaining in tailings	5·5

CONCLUSIONS

The results of the amalgamation tests on both these samples show poor recoveries of the gold values. This may be due to the samples being oxidized, and the gold not being in the best condition for amalgamation. The samples were too small to allow further test work being conducted to prove definitely whether or not better results could be obtained.

The concentration tests showed poor recoveries of the gold values in the concentrates from the Wilfley table, and if the samples are representative, tabling could be eliminated from the milling practice.

The results of the cyanidation tests show that the ore is readily amenable to cyanidation, and that good recoveries can be obtained by this process of treatment.

Test No. 153

A shipment of gold ore was received May 27, 1921, at the Ore Dressing and Metallurgical Laboratories, from the Pan Extension Gold Mines, Limited, Winnipeg, Man. The ore was from their property at Big Rice lake, Man., and consisted of quartz and greenstone, with a small amount of copper and iron sulphides. Free gold was visible, especially in the higher grade sample. The shipping weight was 2,100 pounds, in 27 sacks, one of which was high grade and the remainder lower grade ore.

Test work was desired on this shipment to determine a method of treatment. Amalgamation, concentration, and cyanidation tests were conducted, the results of which are given below.

Samples cut out from the two lots gave the following assays:—

Lower grade lot (26 sacks):.....Copper, 0·60%.....Gold, 2·39 oz. per ton
Higher grade lot (1 sack).....Copper, 0·70%.....Gold, 15·01 oz. per ton

Lower Grade Lot

This lot was crushed to 40 mesh, and a sample cut out for assay. From this crushing some metallics were obtained. It was then put through

an amalgamator and over plates, and the amalgamation tailings concentrated on a Wilfley table, making a concentrate and tailing. The results of this work are contained in the following table:—

Product	Weight pounds	Assay		Content gold oz.	Per cent gold values
		Gold oz.	Cu. per cent		
Bullion from metallics	.....	.....	.....	0·037	1·9
Bullion from amalgam	.....	.....	.....	1·212	62·4
Table concentrate	43·0	9·70	13·65	0·209	10·7
Table tailing	1,939·5	0·50	.....	0·485	25·0
Heads	1,982·5	2·39	.....	1·943 2·369	100·0

A 1,000 gram sample of the table tailing, ground to 100 mesh, was agitated for 8 hours, in 6,000 c.c. of water with 15 grams of sodium cyanide and 10 grams of lime. After agitation the tailing was filtered, washed, dried, sampled, and assayed. The results were as follows:—

Cyanide head	0·50 oz. per ton
Cyanide tailing	0·08 oz. per ton
Recovery by cyanidation	84 per cent, or 21 per cent of the gold values in ore
Remaining in tailing	16 per cent, or 4 per cent of the gold values in ore

Another 1,000 gram sample of the table tailing, ground to 100 mesh, was concentrated in a flotation machine. The following table shows the results obtained:—

Product	Weight, grams	Assay gold, oz.	Content gold, gm.-oz.	Per cent gold value in table tail	Per cent gold value in ore
Concentrate	48	9·24	443·5	80·2	20·1
Middling	162	0·48	77·8	14·1	3·5
Tailing	790	0·04	31·6	5·7	1·4
Table tailing	1,000	0·50	552·4 550·0	100·0	25·0

SUMMARY OF RESULTS

The following tables show the percentage of gold values recovered by the different processes:—

By amalgamation	64·3 p.c.	96·0 %
By table concentration	10·7 p.c.	
By cyanidation	21·0 p.c.	
Remaining in tailing	4·0 p.c.	

By amalgamation	64·3 p.c.	98·6 %
By table concentration	10·7 p.c.	
By flotation	23·6 p.c.	
Remaining in tailing	1·4 p.c.	

Higher Grade Lot

This lot was crushed to 20 mesh. Some metallics were obtained. After sampling, the ore was ground with mercury in a small ball mill, and the mill discharged into an amalgamator, the pulp running over plates to settling boxes. The amalgamation tailing was concentrated on a Wilfley table, making a concentrate and tailing. The results of this work are contained in the following table:—

Product	Weight pounds	Assay		Content gold oz.	Per cent gold values
		Gold oz.	Cu. per cent		
Bullion from metallics	.....	.....	.....	0·035	6·8
Bullion from amalgam	.....	.....	.....	0·410	80·1
Table concentrate	5·0	18·70	3·50	0·047	9·2
Table tailing	36·5	1·12	.....	0·020	3·9
Loss in handling	25·5	.....	.....	.....	.....
Heads	67·0	15·01	.....	0·512 0·503	100·0

A 1,000 gram sample of the table tailing, ground to 100 mesh, was agitated for 8 hours in 6,000 c.c. of water, with 15 grams of sodium cyanide, and 10 grams of lime. After agitation the tailing was filtered, washed, dried, sampled and assayed. The results were as follows:—

Cyanide heads	1·12 oz. per ton
Cyanide tailing	0·06 "
Recovery by cyanidation	94·6 % or 3·7 % of the gold values in ore.
Remaining in tailing	5·4 % or 0·2 % of the gold values in ore.

SUMMARY OF RESULTS

The following table shows the percentage of gold values recovered:—

By amalgamation	86·9	
By concentration	9·2	99·8 %
By cyanidation	3·7	
Remaining in tailing	0·2	

CONCLUSIONS

Drawing our conclusions from the results of the test work on the lower grade lot, as the other was probably a picked sample, or selected from a rich portion of the vein material, we are of the opinion that the ore as shipped to our laboratories is amenable to treatment by amalgamation, table concentration, and cyanidation of the table tailings, or by amalgamation, table concentration, and flotation of the table tailings.

The method of treatment adopted will depend on local conditions. The gold values remaining after amalgamation seem to be associated with the sulphide minerals, and can be recovered in a marketable concentrate by tabling and flotation. This concentrate, besides containing the greater portion of the gold values after amalgamation, will also contain the greater portion of the copper values in the ore.

A simple flow sheet for such an ore would consist of:—

Primary rock crusher, rolls, wet ball mill or stamp battery, amalgamation plates, concentration tables, regrinding ball or tube mill, flotation unit or cyanide plant.

It must be remembered, however, that these conclusions are drawn from the results obtained from the shipment of ore submitted to our laboratories, and are only correct and reliable in so far as the shipment is a representative one of the milling ore from the property.

(11)

CONCENTRATION TESTS ON MOLYBDENITE ORES

C. S. Parsons

Concentration methods and processes for the recovery of molybdenite from its ores, have been fairly well covered by the extensive experimental work carried on in the Ore Dressing and Metallurgical Laboratories during the war years, when there was a brisk demand for the metal for the manufacture of tool-steel and for other munition purposes. Ores from all parts of the Dominion were received and experimental work conducted for the recovery of the molybdenite values. Since that time the industry has been more or less dormant, the production in Canada falling off to practically nothing.

Interest during the year has been revived somewhat, due to the research conducted to find new uses for molybdenite This research has disclosed the fact that when quantities of the metal less than one per cent are used, a beneficial effect is given to most steels.

The experimental work conducted during the year was on ores representing two of the most promising deposits in Ontario and Quebec. It was found that these ores were amenable to concentration, with a high recovery of the molybdenite values in a high grade concentrate. The experimental work is described under the test numbers.

Test No. 154

A shipment of four lots of molybdenite ore, of 200 pounds each, was received at the Ore Dressing and Metallurgical Laboratories, September 15, 1921, from W. E. Joiner, Toronto, Ontario.

These lots represented four types of ore from different points on lots 3 and 4, concession XX, in the township of Cardiff, county of Haliburton, in the vicinity of Wilberforce, Ontario.

These deposits, while being rather low in molybdenite content, have, from the exploration work that has been done on them to date, been reported on as being more or less extensive, and give favourable promise of developing into a source of production of the mineral.

The object of the experimental work was to determine the grade of concentrate and recovery of molybdenite content that could be expected from the ores, and whether they were amenable to concentration by the general methods applied to molybdenite ores.

A sample cut out from each of the four lots gave the following analysis:—

Lot No. 1	MoO_3—0·027 %	MoS_2—0·68 %
2	0·027 %	0·32 %
3	0·025	0·38 %
4		0·12 %

In lot No. 1, the molybdenite flake was of the coarse variety, the ore containing more iron sulphide than the other lots. In lots Nos. 2 and 3 the molybdenite flake was much smaller, being more evenly disseminated through the rock, in which very little iron sulphide was noticeable. In lot No. 4, the rock contained besides molybdenite, an appreciable amount of graphite.

Flotation tests were made on lots Nos. 1, 2, and 3. Lot No. 4 was too low grade to be commercially workable for molybdenite, and for this reason no experimental work was conducted on it.

Table No. I gives the results of the flotation tests. The recoveries indicated include the actual recovery in each product, together with 80 per cent of the molybdenite in the middlings, and the −80 mesh product, which would be recovered in practice by returning them to the rougher cells in closed circuit. From our experience we believe this to be a fair method for the calculation of recoveries.

The tests were conducted in the Callow laboratory flotation cells. Kerosene was used as the collecting oil, and pine oil as the frothing oil. A little lime was added. The cleaner concentrate was screened over an 80 mesh screen, where the floculating effect of the coal oil is taken advantage of to produce a high grade concentrate.

This ore floats readily and gives a high grade concentrate with a good recovery.

TABLE No. I

Test No.	Flotation product	Weight	MoS_2 per cent.	MoS_2 grams	Recovery per cent.	Remarks
1	Concentrate					
	+80 mesh.....	7	93·31	6·53	81·9	Total recovery 91·0 %
	−80 mesh.....	31	2·20	0·68	8·5	assuming that 80 %
	Middling.....	57	0·38	0·22	2·8	of MoS_2 in middling
	Tailing.....	902	0·06	0·54	6·8	could be recovered.
2	Concentrate					
	+80 mesh.....	3	80·25	2·41	75·6	Total recovery 84·1 %
	−80 mesh.....	33	0·58	0·19	5·9	
	Middling.....	86	0·18	0·15	4·7	
	Tailing.....	891	0·05	0·44	13·8	
3	Concentrate					
	+80 mesh.....	4	87·25	3·49	75·4	Total recovery 83·0 %
	−80 mesh.....	39	0·64	0·25	5·4	
	Middling.....	96	0·20	0·19	4·1	
	Tailing.....	872	0·08	0·70	15·1	

Screen test on tailing test No. 1			Screen test on tailing test No. 2		
Mesh	Grams	Per cent	Mesh	Grams	Per cent
+ 35.....	3	0·36	+ 65	2	0·25
− 35 + 48.....	11	1·32	− 65+100	111	13·74
− 48 + 65.....	85	10·22	−100+150	199	24·63
− 65 +100.....	250	30·05	−150+200	240	29·70
−100 +150.....	147	17·67	−200	256	31·68
−150 +200.....	139	16·70			
−200.....	197	23·68			

Screen test on tailing test No. 3

Mesh	Grams	Per cent
+ 65	1	0·13
− 65 +100	176	22·00
−100 +150	225	28·12
−150 +200	120	15·00
−200	278	34·75

Test No. 157

A 100-pound shipment of molybdenite ore was received at the Ore Dressing and Metallurgical Laboratories, July 6, 1921, from Mr. John Bain, Ottawa, Ont.

The ore was from his property at Indian lake, Masham, Que., and consisted of flake molybdenite and heavy iron sulphides in a pyroxenite gangue.

The object of the test work was to determine the grade of concentrate and recovery of the molybdenite content that could be expected, and whether the ore was amenable to treatment by the ordinary methods employed in the concentration of molybdenite ores.

The ore was crushed to pass a 20 mesh screen, and during the process of crushing some flake molybdenite was recovered by screening on a 10 mesh screen.

Material	Weight per cent	Assay MoS_2 per cent.	Contents. Assay × per cent of weight	Per cent of total contents of ore
+10 flake MoS_2	2	64·55	129·10	35·25
−10 mesh	98	2·42	237·16	64·70
Ore	100	3·66	366·26	100·00

The −10 mesh material, assaying 2·42 per cent MoS_2, was ground to pass a 20 mesh screen, and a sample of 1,000 grams was taken, ground in a small ball mill for 15 minutes with 2 c.c. coal oil, 5 drops pine oil, 1 gram lime, and 500 c.c. water. It was then run in a laboratory Callow pneumatic flotation machine, where 5 drops more of pine oil were used, and the necessary water added. The following products were made, the concentrate being screened on an 80 mesh screen:—

Product	Weight per cent	Assay MoS_2 per cent	Contents. Assay × per cent of weight	Per cent of total contents of ore	Per cent of total contents of flotn. feed
−80 concentrate	1·77	89·25	157·97	42·12	65·05
−80 concentrate	3·72	17·00	63·24	16·86	26·04
Cleaner tails	3·33	1·94	6·46	1·72	2·65
Rougher tails	89·18	0·17	15·16	4·05	6·26
	98·00	.....	242·83	64·75	100·00

A screen test of the flotation tailing gave as follows:—

Mesh	Per cent by weight	Cumulative per cent
+ 48	1·2	1·2
− 48 + 65	6·1	7·3
− 65 +100	33·8	41·1
−100 +150	15·5	56·6
−150 +200	21·1	77·7
−200	22·3	100·0
	100·0	.....

In practice the rougher tailing is the only waste product. The other products, namely the −80 concentrate and cleaner tailing, are returned to the circuit and the molybdenite content ultimately recovered from them. The actual recovery expected is represented by the molybdenite content of all products, excepting the rougher tailing. The recovery in the flotation test is, therefore, 93·74 per cent, and if the flake screened out in crushing the ore to 20 mesh is considered, the total recovery of molybdenite in the ore is 95·95 per cent.

The test shows that a concentrate of 90 per cent MoS_2 grade, and a recovery of 95 per cent of the molybdenite content, can be expected from this ore.

(12)

THE RECOVERY OF THE VALUES IN A SILVER-LEAD ORE FROM ARROWHEAD, B.C.

R. K. Carnochan and C. S. Parsons

Test No. 156

A shipment consisting of two lots, one of 20 pounds, representing what was designated as black sands, and the other of 200 pounds, representing what was designated as yellow sands, was received at the Ore Dressing and Metallurgical Laboratories on November 11, 1921.

A sample cut out from each of these lots gave the following analysis:—

Black sands	Gold,	nil
	Silver,	trace
	Lead,	nil
Yellow sands	Gold,	0·52 oz. per ton
	Silver,	4·92 "
	Lead,	4·40%

The analysis of the black sand lot, which consisted of graphitic shale, showed practically no values, so that it was not necessary to conduct any further work on this lot.

The lot designated as yellow sands, which contained lead carbonate and carried values in gold and silver, could be classed as an ore, providing

that the cost of mining and metallurgical treatment was below that of the value of the products which could be recovered from it. As this material showed values of about $16 per ton it had commercial possibilities, and an investigation was undertaken to determine metallurgical methods for the recovery of these values.

Experimental test work was conducted by table concentration to determine what percentage of recovery could be obtained in a concentrate that could be shipped as a smelter product. A concentrate, assaying 5 ozs. gold, 20 ozs. silver, and 40 per cent lead was the best that could be obtained, with a recovery of the gold values of 50 per cent, of the silver values, 20 per cent, of the lead values, 45 per cent. It was found that this treatment must be supplemented by other methods to recover more of the values in the ore.

Test work was conducted by table concentration, producing a concentrate, followed by sulphidizing the table tailing, and flotation for the recovery of more of the values. By this means, a total recovery was obtained of 60 per cent of the gold values, 24 per cent of the silver values, and 38 per cent of the lead values.

A further series of tests was conducted by table concentration, followed by cyanidation of the table tailings. This method of procedure gave the best recoveries. A concentrate was obtained similar to that given above, which with the recovery made by cyanidation, gave a total recovery of the gold values of 95 per cent, of the silver values, 63 per cent, and of the lead values, 45 per cent. This latter process of treatment seems to be the most adaptable to the ore.

Other test work is being conducted, and a report on the investigation will be contained in the Summary Report for next year.

(13)

THE CONCENTRATION OF THE RADIO-ACTIVE MINERAL EUXENITE

R. K. Carnochan

Test No. 159

The occurrence of the radio-active mineral euxenite in the feldspar quarry of lot 13, concession V, of the township of South Sherbrooke, county of Lanark, in the province of Ontario, has been described by Miller and Knight, who visited the property in June, 1917. A report was published in the American Journal of Science, Volume XLIV, September, 1917, and contains a description of the occurrence as seen on that date. The mineral was identified as euxenite by the Imperial Institute, London, England, who gave the analysis of the Canadian occurrence as follows:—

	%
Ta_2O_5	13·89
Nb_2O_5	12·73
TiO_2	27·70
ThO_2	1·34
Ce_2O_3, La_2O_3, etc	0·62
Y_2O_3, etc	25·64
Fe_2O_3	2·63
FeO	0·51

MnO	trace
PbO	0·20
U_3O_8	10·50
CaO	0·09
MgO	0·12
SiO_2	0·74
H_2O, etc	3·00
Total	99·71
Specific gravity	4·99

In the operation of the quarry since that date, more of the mineral has been exposed and shipments have been made to the Department of Mines for experimental work on the concentration of the mineral from the feldspar.

It was found that due to its specific gravity being almost 5, and that of feldspar between 2·5 and 2·75, there was no particular difficulty in making a separation by ordinary gravity means of concentration.

A shipment weighing 1,593 pounds net was received on November 29, 1921, from the Orser-Kraft Feldspar, Limited, Box 266, Perth, Ont. The ore came from the company's property at Maberley, Ont., and consisted of fair sized crystals of black euxenite in red orthoclase feldspar.

It was desired that the euxenite be separated from the feldspar, and that both be recovered in the form of clean products.

A number of specimens were picked out of the ore and the remainder was reduced in a jaw crusher to ¾".

Product	Weight pounds	Per cent of heads
−¾-inch	1,582·00	99·80
Loss	3·25	0·20
Heads	1,585·25	100·00

The −¾" material was crushed in rolls and then screened on 12 mesh by a shaking Ferraris screen, the oversize being fed back to the rolls until it all passed through the screen.

Product	Weight pounds	Per cent of heads
−12 mesh	1,472	92·86
Loss	110	6·94
−¾-inch	1,582	99·80

The −12 mesh material was sized on a Keedy sizer into four sizes: −8+20; −20+42; −42+86; and −86.

Product	Weight pounds	Per cent of heads
− 8 +20	394·0	24·86
−20 +42	535·5	33·78
−42 +86	260 5	16·43
−86	267·0	16·84
Loss	15·0	0·95
−12 mesh	1,472·0	92·86

Each of the four sizes from the Keedy sizer was tabled separately on a large Wilfley table, so as to make a euxenite concentrate and a feldspar tailing of each size. These products were collected as they came from the table in long settling boxes. The water from the tailing box was pumped to a Callow cone during the tabling of all the different sizes, and by this means a certain amount of slime was recovered. After tabling, all products were dried and weighed. The four concentrates produced were very good, being nearly all euxenite, with only a small percentage of feldspar. The tailings were composed of clean feldspar. The slime from the Callow tank was very fine, and drab in colour, and under the microscope was seen to be made up mostly of feldspar, with a little euxenite.

Product	Weight pounds	Per cent of heads	
Concentrate − 8+20	14·00	0·89	
" −20+42	23·50	1·48	
" −42+86	10·50	0·66	3·67
" −86	10·12	0·64	
Tailing − 8+20	370·00	23·34	
" −20+42	504·00	31·79	
" −42+86	244·00	15·39	83·45
" −86	205·00	12·93	
Slimes	42·00	2·65	
Loss	33·88	2·14	
Feed to table	1,457·00	91·91	

The concentrates, after sampling, and a sample representing in proper proportions the combined four sizes of tailings, were shipped to the Orser-Kraft Feldspar, Limited, Perth, Ont.

SUMMARY AND CONCLUSIONS

The euxenite concentrate produced equals 3·67 per cent of the ore treated.

The feldspar tailing product equals 83·45 per cent of the ore treated.

The slimes produced equal 2·65 per cent of the ore treated.

The loss in treatment equals 10·23 per cent of the ore treated.

A good separation of the euxenite from the feldspar can be made. Good clean products can be produced.

Dry crushing and tabling is a very suitable method of treating the ore, as submitted.

IV

OTHER TEST WORK OF THE DIVISION

W. B. Timm

Not included under the heading of investigations, test work was conducted, in some cases partaking of the nature of experimental work, on several shipments, as follows:—

A shipment of 100 pounds, on May 30, of what was supposed to be gold ore from the Grace Mining Co., Ltd., Vermilion Bay, Eagle lake, Ont.

A two-pound sample of screened asbestic sand, from the Windsor Asbestos Co., Ltd., Coleraine, Que., on July 19.

A shipment of 200 pounds of dolomite, on September 19, from the Grasselli Chemical Co., Flower Station, Ont.

A shipment of 2,200 pounds of diatomaceous earth on September 29, from E. A. D. Morgan, K.C., Rawdon, Que.

A shipment of 4,000 pounds of calcite, on November 23, from the Grasselli Chemical Co., Flower Station, Ont.

Supposed Gold Ore from the Grace Mining Co., Vermilion Bay.—The Grace Mining Company, Ltd., made application for the treatment of a few tons of gold bearing ore in the experimental laboratories of the Department of Mines. They were advised of the conditions under which experimental test work was conducted. Conforming to these conditions and advising that they had already a five-stamp battery and other equipment on the property, in which they had made test runs, but due to the varied information given them by a number of their representatives, they shipped 100 pounds of ore for test and check purposes. The shipment as received showed no gold, and only a trace of silver, so that test work as to methods of treatment was not necessary.

Screened Asbestic Sand from the Windsor Asbestos Co., Coleraine.—This sample was received for experimental purposes, to ascertain the amount of total fibre contained, and a report was submitted as follows:—

Weight of sample—900 grams, all used in test. Sample screened into different sizes and each size hand picked, or put over small suction table to remove the fibre. All rock plus twenty mesh was ground in ball mill for five minutes with twenty pounds of balls, screening and suction on sized material was repeated. This operation was repeated on any rock plus twenty mesh remaining, and the fibre removed after each grinding. The following results were obtained:—

Fibre+10 mesh.....	1	gram,	representing.....	0·11%	by weight
Fibre−10 mesh.....	84	grams,	"	9·33	"
Total fibre.....	85	"	"	9·44	"
Rock.....	783	"	"	87·00	"
Loss in handling.....	32	"	"	3·56	"
Total.....	900	"	"	100·00	"

Dolomite, from the Grasselli Chemical Co., Flower Station, Ont.—The object of the work on this shipment was to determine if this dolomite, which was of a pure white variety, could be ground to 250 mesh, and so meet the requirements of the paint and rubber trade. The whole shipment of 200 pounds, with the exception of a few pieces retained for specimens, was crushed in a jaw crusher and rolls and ground in small ball mills. The resultant product gave on screen analysis, the following:—

On 250 mesh	1%
Through 250 mesh	99%

The ground product was shipped back to the company to determine if it met the requirements of the trade.

Diatomaceous or Infusorial Earth from Rawdon, Que.—The work on this shipment was done for the Testing Laboratories of the Department of Public Works. The shipment as received consisted of 2,200 pounds in the natural state and required drying before grinding. The dried material was ground in a conical ball mill equipped with suction, to remove the fine material when disintegrated. The finished product was collected in a collector placed between the suction fan and the mill, and showed on screen analysis that all passed a 100 mesh screen, which was the fineness required. It was shipped to the Testing Laboratories of the Public Works Department.

Calcite from the Grasselli Chemical Co., Flower Station, Ont.—This shipment was from a deposit of pure white calcite in the vicinity of Flower Station. The object of the shipment was to determine the possibilities of grinding to 250 mesh to obtain a product suitable for the paint and rubber trade, and also to determine methods of grinding this material, as it was thought that the pyrite mill of the company could be used for the preliminary crushing, and with certain changes, for fine grinding. In this way the preparation of the material for the market could be accomplished with small additional cost in the way of new equipment.

A series of tests was made on the dry and wet grinding of the calcite. Our equipment, however, was not suitable for dry grinding. Reduction could easily be made to 100 mesh, but the reduction to 250 mesh was impossible on a large scale, with the present equipment. A few hundred pounds was obtained from the test operations, 95 per cent through 250 mesh, and this was shipped to the company to determine if it would meet the requirements of the trade.

A test was made on a ton of the calcite, by wet grinding in a conical mill, charged with pebbles, and in closed circuit with a bowl classifier. No trouble was experienced in grinding the material in this manner. A good tonnage was obtained from the mill, and the overflow of the bowl classifier gave a product practically all through 250 mesh. This product on being dried was slightly off colour, but, no doubt, could be used for certain purposes in the trade.

NOTE.—These tests were conducted by R. K. Carnochan, or under his immediate supervision.

V

ADDITIONAL EQUIPMENT FOR THE LABORATORIES

W. B. Timm

It is of the utmost importance that testing laboratories, where research and experimental work is conducted, be equipped with the latest and most improved type of machinery and apparatus for the metallurgical treatment of ores. To maintain an efficient laboratory in which test work can be carried on, the equipment must be up to date, and for research work it must be in advance of metallurgical practice. It is also of importance that the laboratory be equipped with special machinery and apparatus to cover the wide range of metallurgical problems that are undertaken. It is a simple matter to equip a laboratory to conduct test and research work on one or two particular classes of ore, but it is another problem to meet the needs of the whole industry. For these reasons it has been the policy of the Department to keep the laboratories well equipped, to meet, as far as possible, the requirements of the whole industry.

The equipment of the laboratories consists of two types of machinery and apparatus, namely, the small scale laboratory type for conducting preliminary investigation tests, and complete metallurgical tests on a small scale, and the large scale type of the smallest commercial size machinery, for conducting tonnage check tests, in which the procedure conforms very closely to actual metallurgical practice. On the laboratory type, series of tests are conducted under varying conditions and methods of procedure to determine the procedure most applicable, and when this is determined, a tonnage test is conducted to check results. Small scale tests are conducted on shipments of a few pounds up to 1,000 pounds of ore. Tonnage tests are conducted on shipments from 1,000 pounds up to a carload lot. A chemical laboratory consisting of an analytical and assay laboratory, equipped with modern appliances and apparatus for accurate and rapid determination of analyses and assays of ores and their metallurgical products, is maintained in connection with the testing and research laboratories.

Additional equipment purchased for the laboratories during the year was as follows:—

Standard Asbestos Testing Machine,
Dorr Simplex Classifier,
Magnetic Log Washer,
Roasting Furnace (muffle type).

In connection with the investigation on the milling of asbestos rock, it was necessary to purchase a standard asbestos testing machine, to test the fibre produced from test operations. This is the standard machine used in the asbestos districts, and the fibre produced from the mills is graded to conform to certain tests on this machine.

In connection with the fine grinding of metalliferous ores for flotation work, which in some cases require grinding to 200 mesh, and also for other classification problems, a Dorr simplex classifier equipped with bowl attachment was purchased. This machine was built specially for the testing laboratories, and can be used as an ordinary classifier, or where very fine grinding is necessary, with the bowl attachment. It was placed in closed circuit with a conical ball mill.

A magnetic log washer was purchased and installed for experimental work on the low grade iron ores. This is the latest type of magnetic machine used for the concentration of the low grade magnetic iron ores, having certain advantages for this purpose over the older types of magnetic separators. The addition of this machine to the laboratories makes the equipment fairly complete in the way of conducting experimental work on the beneficiation of the low grade iron ores. This equipment now consists of the necessary crushing and grinding machinery, magnetic cobber, Gröndal magnetic separator, magnetic log washer, laboratory sintering machine, and roasting devices.

A muffle furnace, consisting of four muffles, one above the other, in which the ore to be roasted can be fed to the top muffle and rabbled down to the bottom one where it is discharged, was built for roasting purposes in connection with the experimental work of the laboratories.

During the year certain changes were made in the laboratory equipment to provide for more efficient operation. A Roots blower operated in connection with the pneumatic flotation unit was transferred from the roaster building to the main laboratories. The pipe line leading from the blower to the flotation cells gave considerable trouble, due to vibration. This has now been overcome. An elevator was extended ten feet to elevate the pulp from the discharge of the conical ball mill to the Dorr classifier, and connections made from the classifier for sand oversize to be returned to the mill, and slime overflow to thickening tanks.

A 15 horse-power motor was purchased to drive the machinery in the upper portion of the laboratory and relieve the 25 horse-power motor on the ground floor, which was being overloaded.

A 40 horse-power motor was purchased and installed in the roaster building to operate the machinery and take the place of a locomotive type boiler and engine, which was formerly used for that purpose.

On account of the limited space in the laboratories for new equipment, several pieces of machinery and apparatus were crated and stored. When problems arise which require the use of this machinery, provision is made so that it can be set up again on its old foundation. This is not good policy, but it is the best that can be done under the present conditions, and until more accommodation is provided.

A fitting and repair shop was fitted up with machines and fittings obtained from the Department of Soldiers' Civil Re-Establishment, providing an up to date shop for repairs to the machinery and apparatus of the laboratories.

VI

THE WORK AND INVESTIGATIONS OF THE CHEMICAL LABORATORY

H. C. Mabee

Chemist in Charge

At the beginning of the year the chemical work of the Division was being done in the Mines Branch Chemical Laboratories, Sussex street. The restoration of the temporary frame building containing the chemical laboratories and offices of the Division on Booth street, which were destroyed by fire in February, 1920, was still in progress, and was not completed and the building ready for occupation until the middle of February, when work was resumed in our own laboratories.

Considerably more work was accomplished than in the preceding year, due to the increase in staff by the temporary appointment in July of B. P. Coyne, as associate chemist. A temporary technical officer was appointed as laboratory assistant, and his services were utilized in the chemical laboratories for a portion of the time.

As in past years, the laboratories were employed by the chemical staff in the determination of analyses and assays of ores and experimental test products from the investigations being carried on by the engineering staff of the Division. Assistance was given in the solution of the metallurgical problems covered by the investigations.

The total number of samples analysed and reported on during the year was 838, involving 3,000 chemical determinations. Of this number the largest proportion was from the experimental and research work on the investigations. These samples represented ores and minerals from localities throughout the Dominion, as the following list will indicate:—

Brass slag—8 samples—Refuse product from brass foundry. Determined, tin, antimony, lead, copper, and zinc.

Copper ores—371 samples—Ores from northern Manitoba, Anyox, B.C., and northern Ontario. Determined, copper, copper and nickel and zinc, gold and silver.

Ferro-molybdenum—2 samples—from Mines Branch. Determined, molybdenum, iron, carbon, sulphur, and copper.

Graphite ores—13 samples from Quebec. Determined, carbon.

Graphite shale—4 samples from New Brunswick. Determined, graphitic carbon, volatile and combustible.

Gold ores—33 samples—Ores from Trout lake, B.C., northern Ontario, northern Manitoba, and Quebec. Determined, gold, silver and lead.

Lead-zinc ore—139 samples—from Kimberley, B.C. Determined, lead, zinc, and silver.

Magnesite—23 samples—from Calumet, Que. Determined, lime and magnesia.

Molybdenite—29 samples—from Ontario and Quebec. Determined, molybdenite, molybdic oxide, copper, arsenic, iron, and lime.

Nickel—83 samples—from northern Ontario. Determined, nickel.

Nickel-copper—71 samples—ores from Sudbury, Ont. Determined, nickel and copper.

Platinum—8 samples—from the Cariboo district, B.C. Determined, platinum and gold.

Silver—54 samples—slimes from Cobalt ores and ores from Cape Breton. Determined, silver.

Besides the regular assays and chemical analyses performed on the numerous ores and minerals as indicated above, special investigation was also carried on in the case of some of the more complex ores, with the view of ascertaining the presence of any rare elements or precious metals.

The Recovery of the Metals of the Platinum Group from Placer Concentrates

Part I

In 1919 an investigation was undertaken on a small shipment of placer concentrates for the recovery of the metals of the platinum group. The experimental work consisted of trying a new method for the amalgamation of the platinum group metals, as follows:—

The dressed plates were sprinkled with finely powdered zinc amalgam, made by pouring molten zinc into mercury, and grinding the amalgam thus produced to powder. The platinum sands were ground to 150 mesh and fed over the plates in the form of a wet pulp containing a weak solution of copper sulphate and sulphuric acid; an electro-chemical action takes place, forming a permanent coating of copper on the platinum, which in passing over the plates is held by the mercury. Due to the limited supply of material submitted and the small quantity of platinum in the sample, the results obtained were doubtful as to accuracy. The results were as follows:—

Pulp before amalgamation	Au. 0·10 oz.;	Pt. 0·003 oz.
Pulp after amalgamation	Au. 0·06 oz.;	Pt. 0·001 oz.
Extraction	Au. 40 p.c.	Pt. 66 p.c.

A larger sample of 450 pounds of concentrates from the same locality was received, and in the preparation of a sample cut out from the shipment for assay, it was found that all the metals of the platinum group were caught on a 100 mesh screen. The assay of this sample was —Au. 0·41 oz; Pt. 0·032 oz. —100 mesh material after removal of metallics —Au. 0·39 oz.; Pt. trace.

The results of this assay showed that the metals of the platinum group in these particular concentrates were free, and occurred in the form of fine flake-like particles coarser than 100 mesh. This finding was substantiated by further work. In order to obtain some material high enough in platinum content so that the above method could be tried for the recovery of the platinum group metals, the remainder of the shipment was ground in ball mills for two hours and screened on a 100 mesh screen. This gave the following products:—

Product	Weight lbs.	Au. oz. per ton	Pt. oz. per ton
+100 mesh	107	0·90	0·105
−100 mesh	278	0·10	trace

In the assay of the + 100 mesh material from a sample of 1,518 grams ground to pass 100 mesh, 10 grams of metallics were caught on the screen, which lot was assayed separately from the material passing through the screen. This assay gave when figured to the ton of head sample, of +100 mesh product:—

Metallics	Au. 0·48 oz.	Pt. 0·101 oz.
−100 mesh	Au. 0·42 oz.	Pt. 0·005 oz.
Total	Au. 0·90 oz.	Pt. 0·106 oz.

These results check the former ones, and prove that the values of the platinum group metals in this concentrate can be recovered as metallics by grinding to 100 mesh.

The +100 mesh product which shows values in gold of 0·90 oz. per ton, and in platinum group metals of 0·105 oz. per ton, will be used for experimental work in recovering the values by the method described above. This work will be conducted next year.

Part II

Experimental work was conducted by making fusion tests on the placer concentrates, which consist of black sands or iron oxides, chiefly the magnetic oxide containing gold and metals of the platinum group. In some cases the precious metals are locked up in the metallic oxides, and the usual methods of amalgamation have failed, even on fine grinding, to make a satisfactory recovery of these metals.

A series of experiments was undertaken along similar lines to those employed in the regular crucible assay, namely, of fusing and slagging off the iron content and gangue, and collecting the precious metals in a lead button by the addition of galena to the charge. Several mixtures were employed, using a quantity of limestone, silica, and charcoal as fluxing materials, a sufficient quantity of galena containing about 60 per cent metallic lead, and about 50 ozs. of silver. It was hoped to melt this mixture at a fairly high temperature to form a liquid slag, and produce metallic lead for the collection of the precious metal values. The fusions were made in a Case muffle furnace, but it was found that a much higher temperature was required than it was possible to obtain in this furnace, to produce a liquid slag. Further experimental work along this line will be conducted in other types of furnaces.

FUELS AND FUEL TESTING DIVISION

I

LIGNITE CARBONIZATION

CONTINUATION AND REVIEW OF SMALL SCALE EXPERIMENTS

J. H. H. Nicolls and Harold Kohl

Introductory.—The main purposes of these small scale experiments on the lower grade Canadian coals were (*a*) to obtain information concerning the nature of the carbonized residues produced by their destructive distillation; (*b*) by means of this information to indicate whether the increase in heating value of any one of the coals would warrant its carbonization, and, if so, at what temperature it should be carbonized.

In the Mines Branch Summary Reports[1] for the years 1917 to 1920 the account of the carbonization experiments on Canadian fuels was each year brought up to date. The earlier reports dealt mainly with the lignite coals from the Estevan area in Saskatchewan, upon which both small and large scale laboratory experiments were carried out. Subsequent reports described small scale experiments on the lower grade Alberta coals, and a sample of peat from Alfred, Ontario.

The present report describes small scale carbonization experiments upon the following five Canadian coals:—

No. 1665—From McCormack mine, Castor, Alta.
No. 1790—From Midland collieries, Drumheller, Alta.
No. 1795—From Sinclair mine, Hanna, Alta.
No. 1824—From Canada West Coal Co., Taber, Alta.
No. 1875—From Goodlands, Man.

The account of these experiments is followed by a resumé of all the small scale laboratory experiments referred to above. In the resumé comparisons are drawn between the lignites from Saskatchewan and those from Alberta, which in turn are compared with Alfred peat.

Experimental.—The fuels were carbonized in the form of small briquettes. A small quartz crucible was filled with these and placed inside a retort, which was heated to the desired temperature in a bath of molten lead. In order that a reducing atmosphere should be present throughout the test, the air in the retort was displaced by coal gas at the beginning of the operation. The outlet tube of the retort dipped just below the surface of a water seal, from which the gas passed to the outer air, thus ensuring that carbonization took place practically at atmospheric pressure. The apparatus is more fully described in the Mines Branch Summary Reports for 1918 and 1920.

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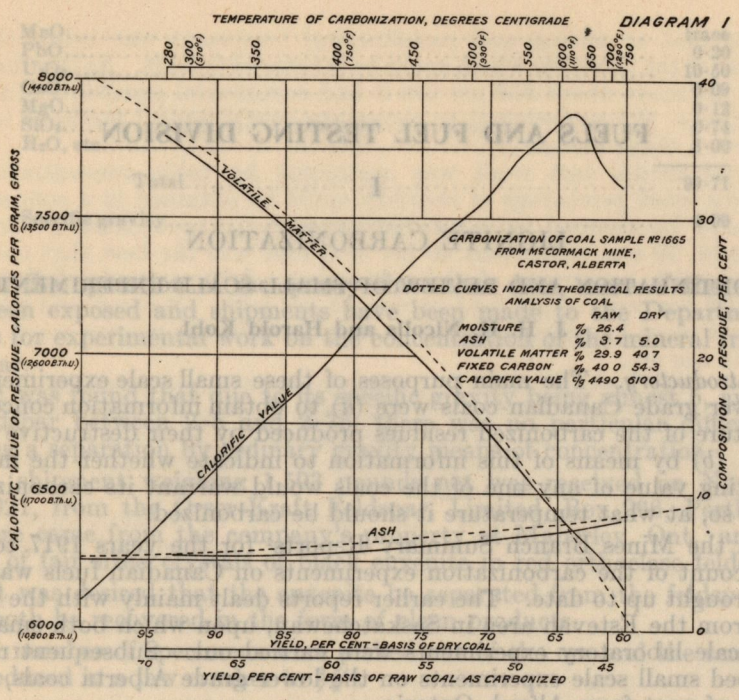
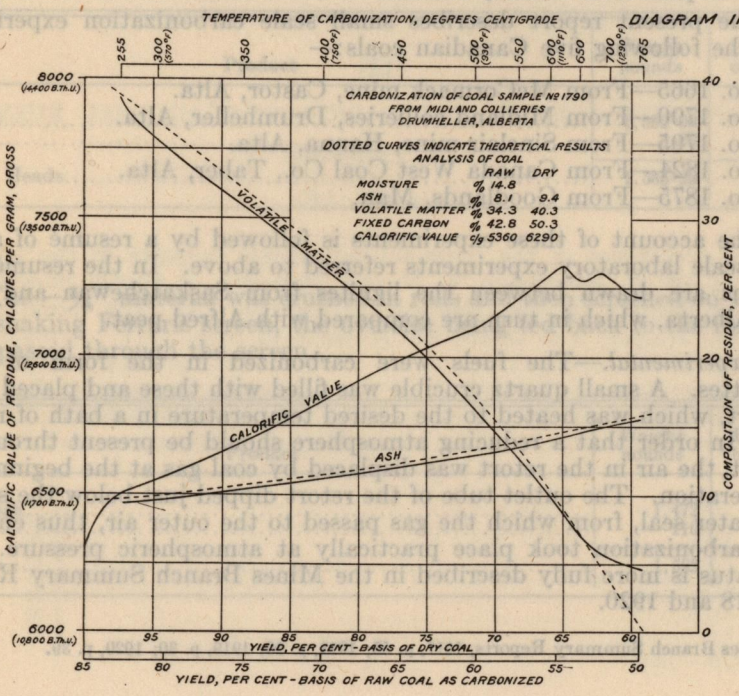
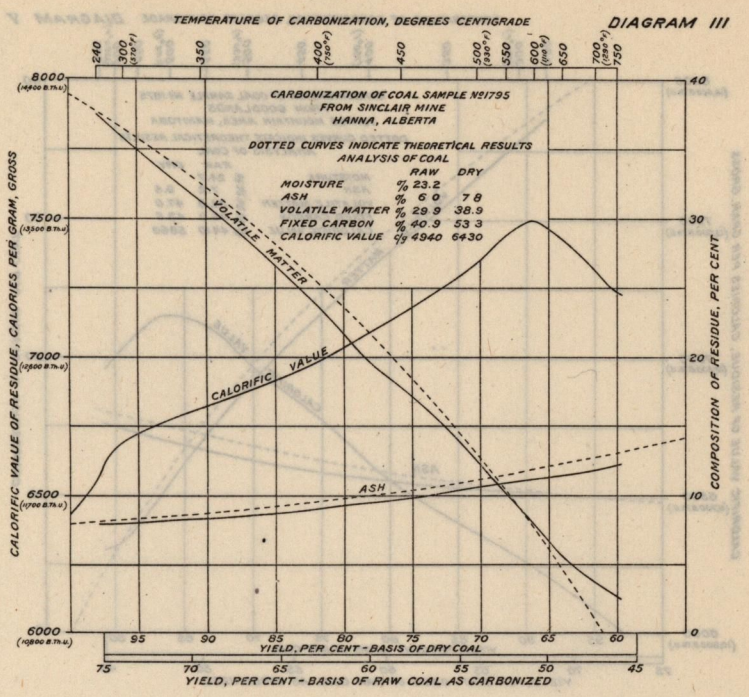
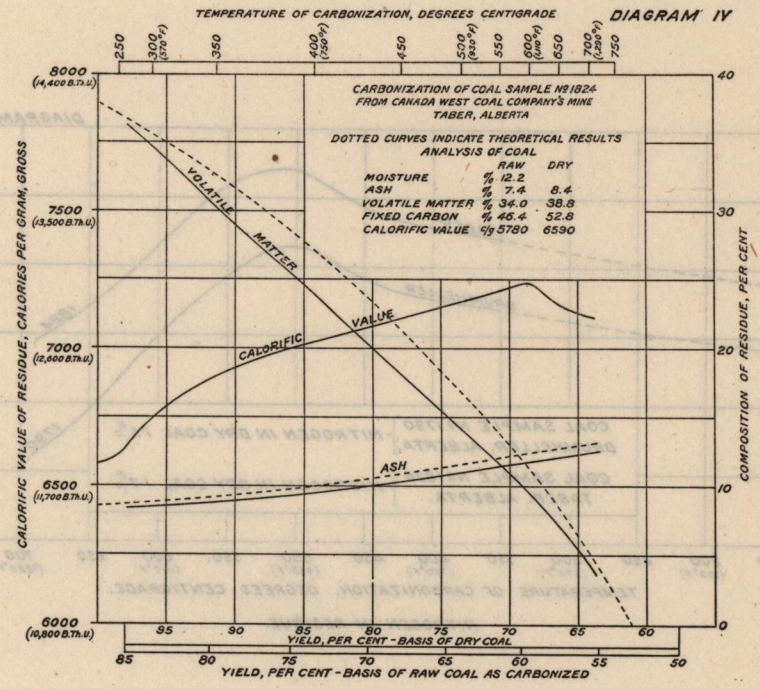
[1]Mines Branch Summary Reports, 1917, p. 55; 1918, p. 90; 1919, p. 30; 1920, p. 39.


TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE
DIAGRAM I
280 300 (570°F) 350 400 (750°F) 450 500 (930°F) 550 600 (1110°F) 650 700 (1290°F) 750
8000 (14400 B.Th.U)
7500 (13500 B.Th.U)
7000 (12600 B.Th.U)
6500 (11700 B.Th.U)
6000 (10800 B.Th.U)
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM, GROSS
COMPOSITION OF RESIDUE, PER CENT
40
30
20
10
0
VOLATILE MATTER
CALORIFIC VALUE
ASH
CARBONIZATION OF COAL SAMPLE № 1665
FROM McCORMACK MINE,
CASTOR, ALBERTA
DOTTED CURVES INDICATE THEORETICAL RESULTS
ANALYSIS OF COAL
RAW DRY
MOISTURE % 26.4
ASH % 3.7 5.0
VOLATILE MATTER % 29.9 40.7
FIXED CARBON % 40.0 54.3
CALORIFIC VALUE Cl/g 4490 6100
95 90 85 80 75 70 65 60
YIELD, PER CENT - BASIS OF DRY COAL
70 65 60 55 50 45
YIELD, PER CENT - BASIS OF RAW COAL AS CARBONIZED


TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE
DIAGRAM II
255 300 (570°F) 350 400 (750°F) 450 500 (930°F) 550 600 (1110°F) 650 700 (1290°F) 765
8000 (14400 B.Th.U)
7500 (13500 B.Th.U)
7000 (12600 B.Th.U)
6500 (11700 B.Th.U)
6000 (10800 B.Th.U)
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM, GROSS
COMPOSITION OF RESIDUE, PER CENT
40
30
20
10
0
VOLATILE MATTER
CALORIFIC VALUE
ASH
CARBONIZATION OF COAL SAMPLE № 1790
FROM MIDLAND COLLIERIES
DRUMHELLER, ALBERTA
DOTTED CURVES INDICATE THEORETICAL RESULTS
ANALYSIS OF COAL
RAW DRY
MOISTURE % 14.8
ASH % 8.1 9.4
VOLATILE MATTER % 34.3 40.3
FIXED CARBON % 42.8 50.3
CALORIFIC VALUE Cl/g 5360 6290
95 90 85 80 75 70 65 60
YIELD, PER CENT - BASIS OF DRY COAL
85 80 75 70 65 60 55 50
YIELD, PER CENT - BASIS OF RAW COAL AS CARBONIZED


TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE
DIAGRAM III
CARBONIZATION OF COAL SAMPLE №1795
FROM SINCLAIR MINE
HANNA, ALBERTA
DOTTED CURVES INDICATE THEORETICAL RESULTS
ANALYSIS OF COAL
RAW DRY
MOISTURE % 23.2
ASH % 6 0 7 8
VOLATILE MATTER % 29 9 38.9
FIXED CARBON % 40.9 53 3
CALORIFIC VALUE % 4940 6430
VOLATILE MATTER
CALORIFIC VALUE
ASH
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM, GROSS
COMPOSITION OF RESIDUE, PER CENT
YIELD, PER CENT - BASIS OF DRY COAL
YIELD, PER CENT - BASIS OF RAW COAL AS CARBONIZED


TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE
DIAGRAM IV
CARBONIZATION OF COAL SAMPLE №1824
FROM CANADA WEST COAL COMPANY'S MINE
TABER, ALBERTA
DOTTED CURVES INDICATE THEORETICAL RESULTS
ANALYSIS OF COAL
RAW DRY
MOISTURE % 12.2
ASH % 7.4 8.4
VOLATILE MATTER % 34.0 38.8
FIXED CARBON % 46.4 52.8
CALORIFIC VALUE % 5780 6590
VOLATILE MATTER
CALORIFIC VALUE
ASH
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM, GROSS
COMPOSITION OF RESIDUE, PER CENT
YIELD, PER CENT - BASIS OF DRY COAL
YIELD, PER CENT - BASIS OF RAW COAL AS CARBONIZED

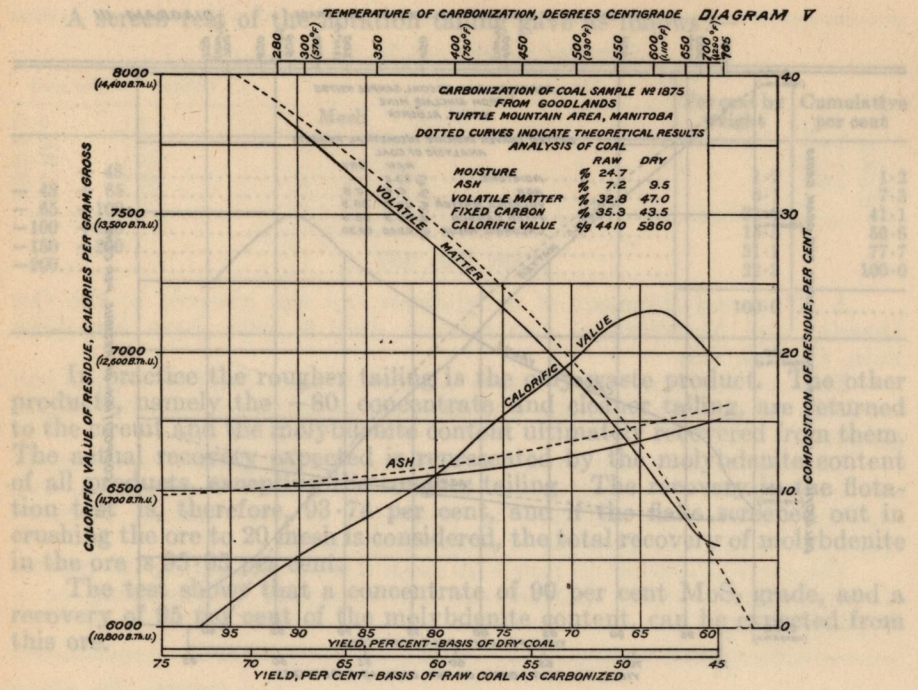


DIAGRAM V
TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE
CARBONIZATION OF COAL SAMPLE Nº 1875
FROM GOODLANDS
TURTLE MOUNTAIN AREA, MANITOBA
DOTTED CURVES INDICATE THEORETICAL RESULTS
ANALYSIS OF COAL
RAW DRY
MOISTURE % 24.7
ASH % 7.2 9.5
VOLATILE MATTER % 32.8 47.0
FIXED CARBON % 35.3 43.5
CALORIFIC VALUE cy 4410 5860
VOLATILE MATTER
CALORIFIC VALUE
ASH
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM, GROSS
COMPOSITION OF RESIDUE, PER CENT
YIELD, PER CENT - BASIS OF DRY COAL
YIELD, PER CENT - BASIS OF RAW COAL AS CARBONIZED

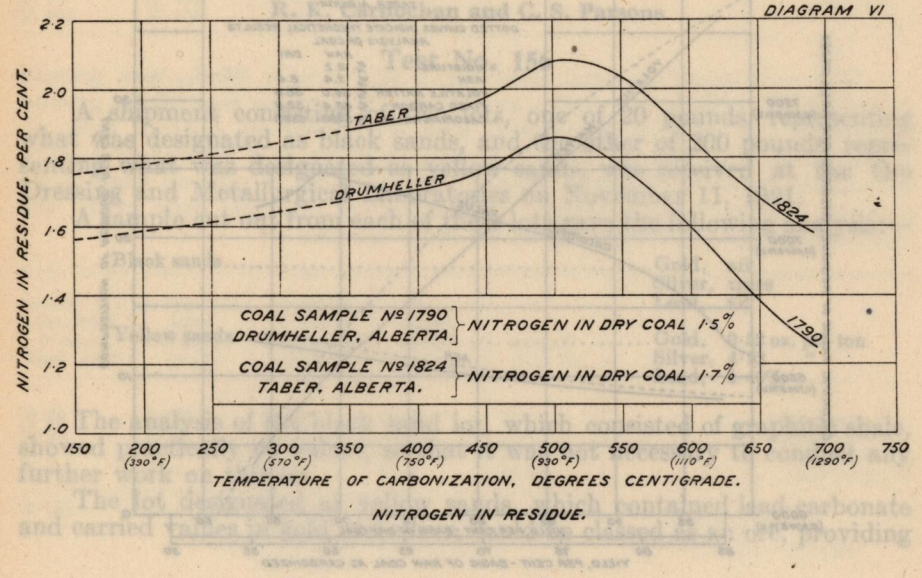


DIAGRAM VI
NITROGEN IN RESIDUE, PER CENT.
TABER
DRUMHELLER
1824
1790
COAL SAMPLE Nº 1790
DRUMHELLER, ALBERTA.
NITROGEN IN DRY COAL 1·5%
COAL SAMPLE Nº 1824
TABER, ALBERTA.
NITROGEN IN DRY COAL 1·7%
TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE.
NITROGEN IN RESIDUE.

Discussion of Results.—The results of the experiments are given in Tables I to V, and graphically indicated in Diagrams I to V. The curves were plotted from the experimental results, and to a certain extent rounded off, and from these rounded curves the figures in the tables were obtained. In each case the fuel was carbonized in the raw state, but, for the sake of uniformity, the curves were plotted against the yields calculated to be obtainable from the dried fuel.

As was found to be the case in nearly all previous series, the determined percentages of ash and volatile matter in the carbonized residues were lower than the calculated values. These were calculated from the analyses of the original coals and the determined yields of residue. As was mentioned in a previous report,[1] freshly carbonized residue will absorb air and moisture rapidly, so that the yield calculated from its weight after cooling would be higher than the correct value. On this account also the composition of the residue as analysed would differ from that of the residue as removed from the lead bath, in that it would contain more volatile matter and slightly less ash and fixed carbon. This, however, would not account for the divergence between the percentages of volatile and ash as determined and those computed from the yield calculated from the weight of residue.

In referring to the diagram for the Castor coal it will be observed that the calorific value rises to an unusual extent, the increase at the optimum temperature, 620° C., being almost thirty per cent. This is far greater than the rise for any other coal, and almost as high as that for peat. Furthermore, the curve is exceptionally regular, and not bowed out in places as is the case with most of the other coals. This indicates that decomposition occurs uniformly with rise of temperature. Unfortunately this coal was slightly more weathered than most of the other coals carbonized, as it was kept in a finely ground condition (though in a sealed jar) for one year before the tests were made. This however would probably have little or no effect on the results of carbonization, unless it should slightly lower the calorific value.

In the case of the Drumheller coal the calorific value curve was found to be abnormal above the optimum temperature, 620° C., in that it rose into a second peak. It followed a normal course, dropping until 650° C. was reached, but then it rose again until 685° C. was attained, after which it again dropped. The maximum temperature reached was 765° C., up to which point the curve dropped gradually from the second peak. Owing to the abnormal character of the curves, check carbonizations were made at intervals of 10 or 15 degrees between 600° and 700° C., some of these after two or three months had elapsed since the first series. As a final and very severe test, the following procedure was adopted. The coal was carbonized at 650° C. for two hours, according to the regular procedure, and part of the residue removed for determination of calorific value; the retort was then closed, and carbonization continued for two hours at 685° C., after which the calorific value of the remaining residue was determined. Owing to adverse conditions, such as opening the retort between the first and second carbonizations, it was not anticipated that

[1]Mines Branch Summary Report, 1918, pp. 91 and 92.

the increase in calorific would be as large as when separate tests were made. It was therefore very satisfactory to find that the average increase for several experiments was 15 to 20 calories. As a result of these tests it was assumed that at 650° C. there was a change in the nature of the volatile matter given off, its calorific value being decreased, with a corresponding increase in that of the residue.

It had been observed that, after carbonization at 500° C. and higher temperatures, there was, on opening the retort, a stronger odour of ammonia from this Drumheller coal than from the other fuels carbonized. Therefore, as the first step in the investigation of the abnormality above referred to, determinations were made of the percentages of nitrogen left in the residues at various temperatures. These are indicated in Diagram VI, where they are compared with the percentages remaining in the Taber coal, which behaved normally on carbonization. While these results are incomplete without the ammonia yield, they indicate that the nitrogen content is not an important factor in the case under consideration. Investigation of the abnormal behaviour of this coal is to be continued, and the results published at some future date.

The curves for the remaining three coals, particularly that from Hanna, are apparently normal. Unless subsequent investigations should reveal a valuable yield of by-products, there would probably be no advantage in carbonizing the Taber coal. On the other hand, it would most likely be economically feasible to carbonize the very low grade coal from Goodlands, Manitoba, since the calorific value of this fuel is increased by 22 per cent on carbonization.

The above experiments complete the twelve series of small scale laboratory tests as originally planned. In order to compare the behaviour of the several coals on carbonization, Diagrams VII to XVIII have been prepared. In these diagrams the yields, calculated to be obtainable from the dried fuels, and calorific values have been plotted against temperatures.

It will be observed that the ash content of the dried fuel is included in the legend for each curve, for the following reason. In numerous instances it has been found that, in the case of samples from the same colliery or for coals similar in composition, increase in ash content decreases not only the calorific value of the fuel as it stands, but also decreases its calorific value when calculated on an ash and moisture free basis. It should therefore be borne in mind when comparing the curves that the percentages of ash in the fuels vary from 5·0 to 11·6, which in itself would account for a considerable discrepancy. Furthermore, in considering a fuel with a high ash content, it should be realized that a low ash fuel from the same source would probably have an even greater calorific value that would be indicated by calculation.

Table VI shows the yields, calorific values and percentage increases in calorific value, at 400° C. and the optimum temperature for each of the twelve fuels carbonized. The temperature 400° C. was originally selected for the reason that the calorific curve for peat flattened very considerably at that point; it was subsequently retained as being conveniently intermediate between the point of initial decomposition and the optimum temperature. By the latter term is meant the temperature at which the carbonized residue has its highest calorific value.

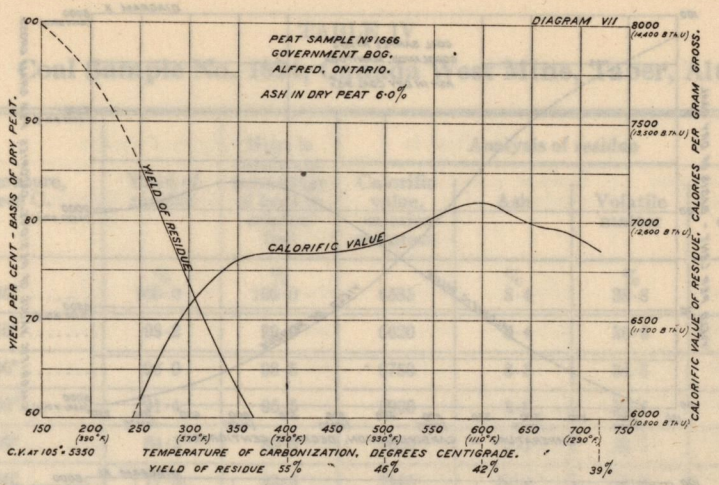


DIAGRAM VII
PEAT SAMPLE Nº 1666
GOVERNMENT BOG.
ALFRED, ONTARIO.
ASH IN DRY PEAT 6·0%
YIELD OF RESIDUE
CALORIFIC VALUE
YIELD PER CENT - BASIS OF DRY PEAT.
CALORIFIC VALUE OF RESIDUE. CALORIES PER GRAM GROSS.
C.V. AT 105° = 5350
TEMPERATURE OF CARBONIZATION. DEGREES CENTIGRADE.
YIELD OF RESIDUE 55% 46% 42% 39%

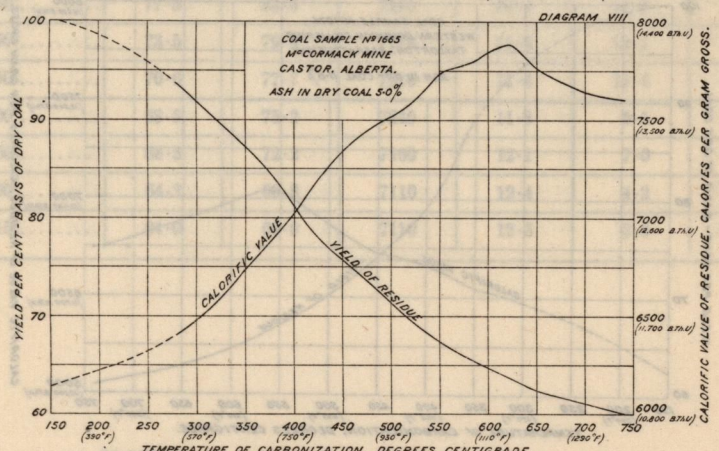


DIAGRAM VIII
COAL SAMPLE Nº 1665
McCORMACK MINE
CASTOR, ALBERTA.
ASH IN DRY COAL 5·0%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL
CALORIFIC VALUE OF RESIDUE. CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION. DEGREES CENTIGRADE

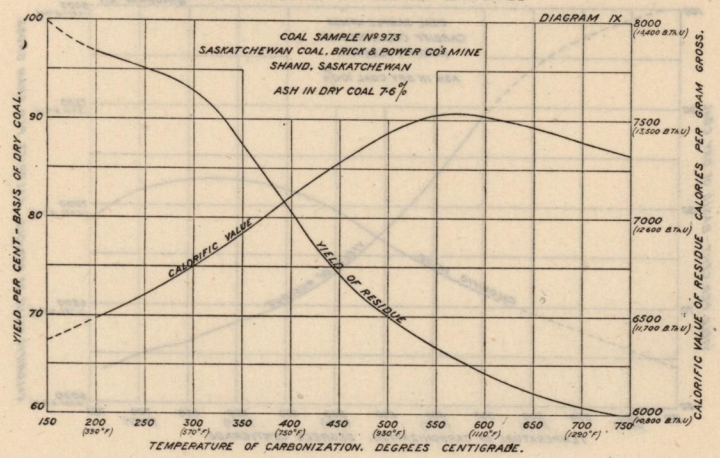


DIAGRAM IX
COAL SAMPLE Nº 973
SASKATCHEWAN COAL, BRICK & POWER Co's MINE
SHAND, SASKATCHEWAN
ASH IN DRY COAL 7·6%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL.
CALORIFIC VALUE OF RESIDUE CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION. DEGREES CENTIGRADE.

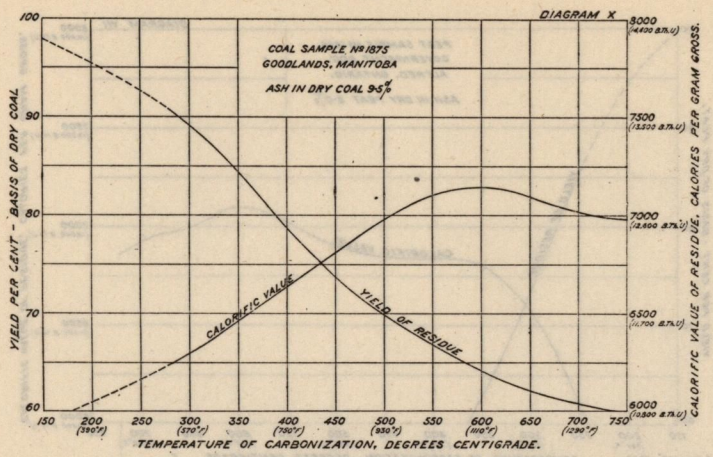


DIAGRAM X
COAL SAMPLE No 1875
GOODLANDS, MANITOBA
ASH IN DRY COAL 9·5%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE.

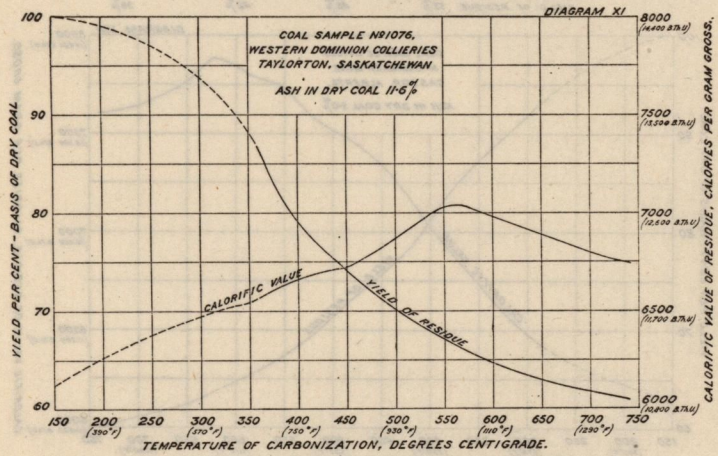


DIAGRAM XI
COAL SAMPLE No 1076,
WESTERN DOMINION COLLIERIES
TAYLORTON, SASKATCHEWAN
ASH IN DRY COAL 11·6%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE.

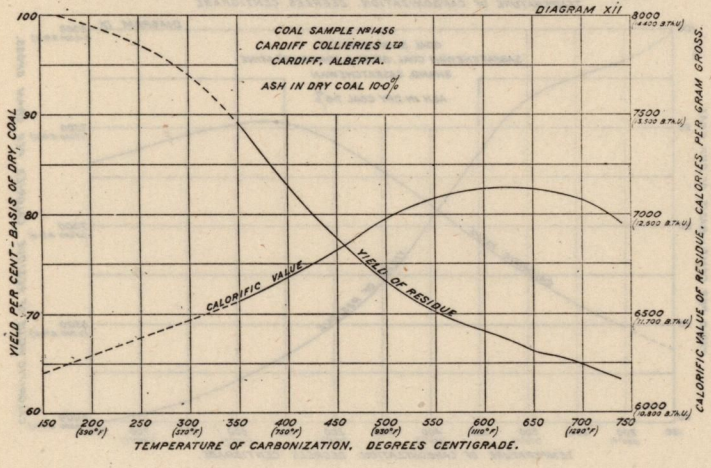


DIAGRAM XII
COAL SAMPLE No 1456
CARDIFF COLLIERIES LTD
CARDIFF, ALBERTA.
ASH IN DRY COAL 10·0%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE.

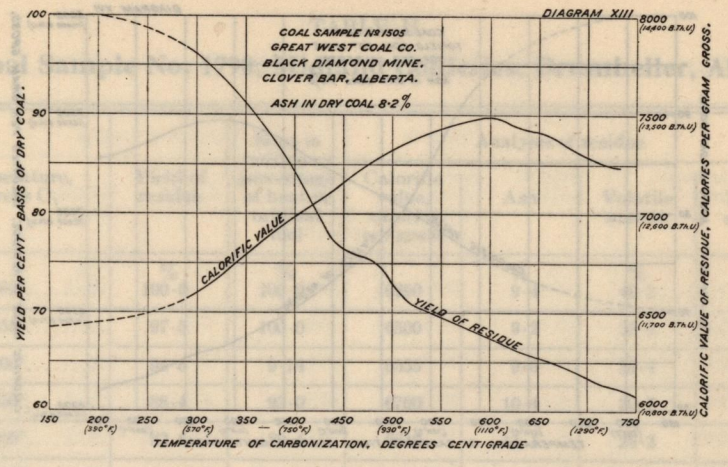


DIAGRAM XIII
COAL SAMPLE No 1505
GREAT WEST COAL CO.
BLACK DIAMOND MINE.
CLOVER BAR, ALBERTA.
ASH IN DRY COAL 8·2%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL.
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE

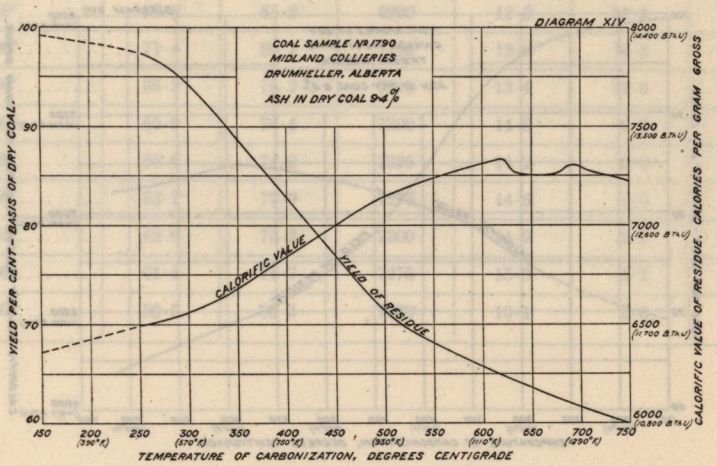


DIAGRAM XIV
COAL SAMPLE No 1790
MIDLAND COLLIERIES
DRUMHELLER, ALBERTA
ASH IN DRY COAL 9·4%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL.
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM GROSS
TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE

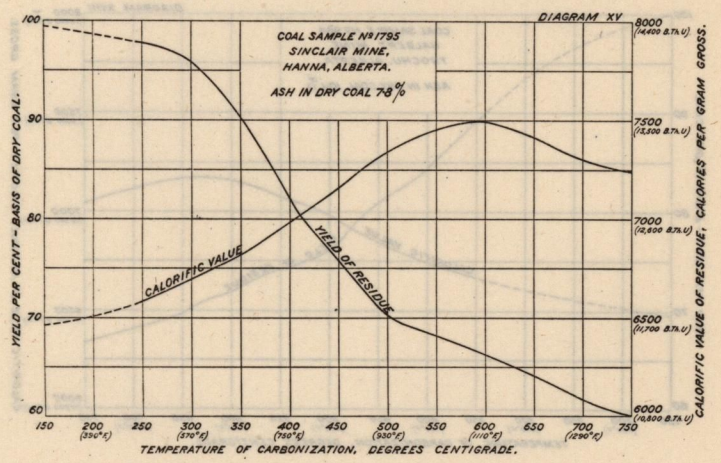


DIAGRAM XV
COAL SAMPLE No 1795
SINCLAIR MINE,
HANNA, ALBERTA.
ASH IN DRY COAL 7·8%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL.
CALORIFIC VALUE OF RESIDUE, CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION, DEGREES CENTIGRADE.

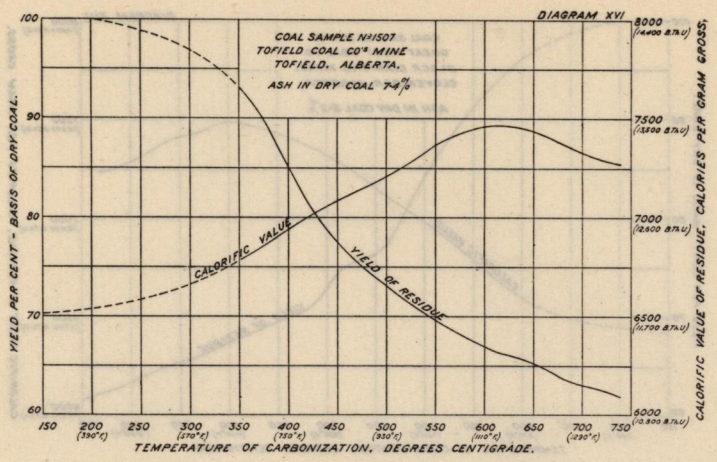


DIAGRAM XVI
COAL SAMPLE N°1507
TOFIELD COAL CO.ª MINE
TOFIELD, ALBERTA.
ASH IN DRY COAL 7·4%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL.
CALORIFIC VALUE OF RESIDUE. CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION. DEGREES CENTIGRADE.

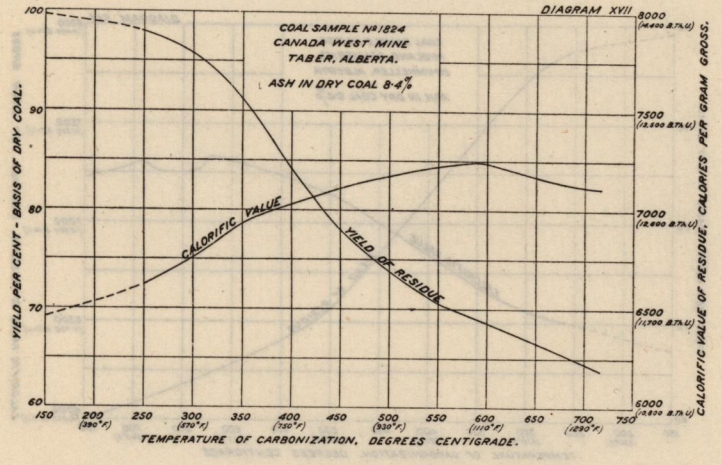


DIAGRAM XVII
COAL SAMPLE N°1824
CANADA WEST MINE
TABER, ALBERTA.
ASH IN DRY COAL 8·4%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL.
CALORIFIC VALUE OF RESIDUE. CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION. DEGREES CENTIGRADE.

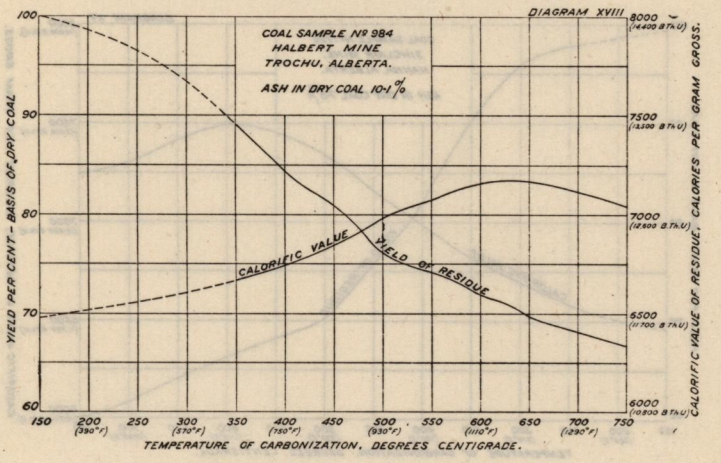


DIAGRAM XVIII
COAL SAMPLE N° 984
HALBERT MINE
TROCHU, ALBERTA.
ASH IN DRY COAL 10·1%
CALORIFIC VALUE
YIELD OF RESIDUE
YIELD PER CENT - BASIS OF DRY COAL
CALORIFIC VALUE OF RESIDUE. CALORIES PER GRAM GROSS.
TEMPERATURE OF CARBONIZATION. DEGREES CENTIGRADE.

The percentage increases in calorific value for the Alfred peat and the Castor coal are exceptionally high, while those for the Shand and Goodlands coals also form a distinct group. Generally speaking, the Alberta coals give a considerably lower percentage increase than those from Saskatchewan and Manitoba. The Taylorton (Saskatchewan) coal seems to be out of place among the Alberta coals, but this misplacement may be due to its high ash content.

In Table VII is given a comparison between the percentage increases in calorific value of the carbonized residue calculated on the basis of the dry fuel and that calculated on the raw fuel basis. In the latter instance the proportions of the increase due respectively to oven drying and to carbonizing are indicated. Generally speaking, the moisture contents indicated represent the average for several samples from the same mine; this does not, however, hold for the Drumheller coal, where the moisture indicated is at least 3 per cent lower than the average. The Goodlands coal, when carbonized, contained 24·7 per cent of water, but, as the average moisture content of samples from the same deposit was 31·7 per cent, the latter figure was used in the table. The moisture content indicated for the Alfred peat approximates that for the air-dried fuel, whereas the peat as excavated from the bog contains as much as 90 per cent of water. It is to be noted that in this table all the calorific values are expressed as British Thermal Units per pound.

In Table VIII is shown the available heat in the residue as a percentage of the available heat in the original fuel. This is shown at various temperatures, for each of the twelve fuels, and is identical with the "Thermal Efficiency" referred to in a previous report.[1] It may perhaps be best explained by the following calculation:—

Assume that 50 grams of coal, with a calorific value of 6,000 calories per gram, yields a residue of 40 grams with a calorific value of 7000 calories. Then the available heat percentage is:—

$$\frac{7000 \times 40 \times 100}{6000 \times 50} = 93{\cdot}3 \text{ per cent.}$$

Diagrams XIX and XX show graphically these available heat percentages for six representative fuels, and, while there are one or two divergences, it will be noted that most of the curves follow approximately the same path. The curves are plotted against temperature in one instance, and yield in the other. Except in the case of peat, the differences obtained are not of much account.

Summary Remarks.—The principal object of these small scale experiments has been brought about; namely, the determination of the changes produced in the calorific value of certain fuels by carbonization at various temperatures and at atmospheric pressure. These changes are indicated by diagrams and tables for the five coals most recently examined. In addition, a resumé is given of small scale experiments upon eight coals from Alberta, two from Saskatchewan, one from Manitoba, and a sample of Ontario peat. Comparisons are drawn between these by means of a series of diagrams and tables.

[1] Mines Branch Summary Report, 1918, p. 93.

DIAGRAM XIX

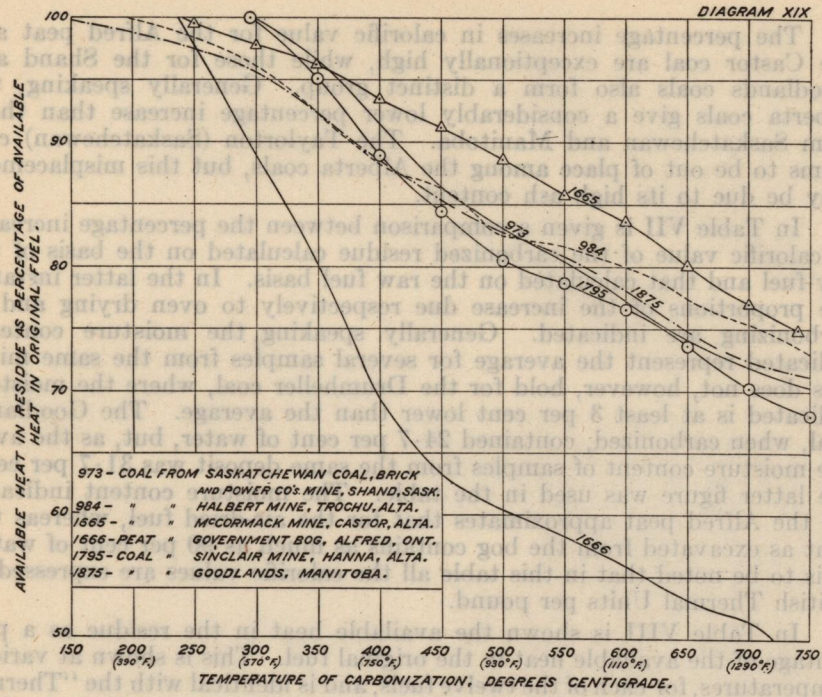
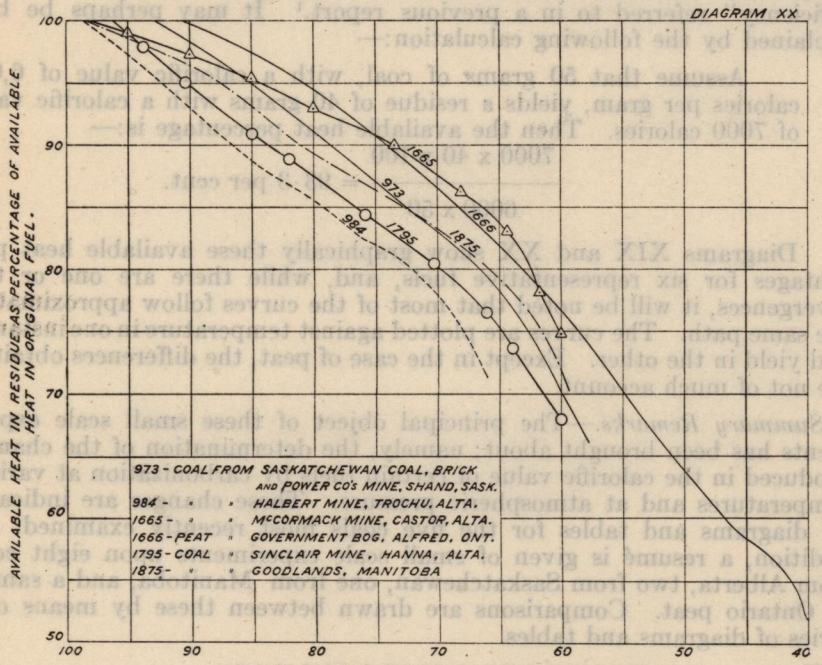


DIAGRAM XX



The lignite coals dealt with in this report may be divided into two main classes, (1) those from Manitoba and Saskatchewan, and (2) lignites of a somewhat higher grade, or "domestic" coals from Alberta. It should be noted that, while sub-bituminous and higher grade coals also occur in Alberta, it is only with the "domestic" coals that this report is concerned.

The coals from Manitoba and Saskatchewan contain over 30 per cent moisture, and disintegrate on exposure, which detracts very considerably from their value when in the raw state. Carbonization of these low grade coals not only remedies this deficiency, but also raises their calorific value from 4200 calories (7500 B.Th.U.) in the raw state, or 6100 calories (11000 B.Th.U.) in the dry state, to 7200 calories (13000 B.Th.U.), increases respectively of 73 per cent and 27 per cent. They therefore appear to require a heat treatment before they can be used other than locally as a household fuel.

The Alberta coals, on the other hand, contain from 15 per cent to 25 per cent moisture, and do not disintegrate quite so readily as those of the lower grade. Carbonization of these, excepting the Castor coal, raises their fuel value on an average 40 per cent (25 per cent to 53 per cent) above that of the raw fuel, or 15 per cent (10 per cent to 17 per cent) above that of the dried fuel. Therefore the question of by-products would probably have to be taken into account when considering heat treatment of any of these coals.

It will be noted that (as shown in Table VII) over 50 per cent of the increase in calorific value caused by carbonization, calculated on the raw fuel basis, is due to loss of moisture. This holds good except for the peat and the Drumheller coal, which do not represent the average run of raw fuel, and for the Castor coal which appears to be abnormal. The increases due to loss of moisture range from 44 per cent to 65 per cent of the total increases in calorific, but vary too much to serve as a basis of division between the different coals. They are, however, of interest in showing the comparative values of the coals for any such use as powdered fuel, where a low moisture content is desirable.

With respect to the maximum increase in calorific value of the carbonized residue, the eleven coals may be roughly classed as follows:—

(*a*) Nos. 1875, 973 and 1076—from Goodlands, Manitoba, and Shand and Taylorton, Saskatchewan—show an average increase in calorific value of 74·7 per cent on the raw basis, and 19·6 per cent on the dry basis. Their calorific values average as follows: raw fuel, 4140 calories (7460 B.Th.U.), dried fuel, 6050 calories (10890 B.Th.U.), carbonized residue, 7240 calories (13030 B.Th.U.).

(*b*) No. 1665—from Castor, Alberta—shows an increase in calorific value of 75·6 per cent on the raw basis, or 29·2 per cent on the dry basis. It has the following calorific values: raw fuel, 4490 calories (8080 B. Th. U.), dried fuel, 6100 calories (10980 B.Th.U.), carbonized residue, 7880 calories (14180 B.Th.U.). In this case the low ash content of the sample may be significant.

(*c*) Nos. 1456, 1505, 1790, 1795, 1507—Alberta coals from Cardiff, Clover Bar, Drumheller, Hanna and Tofield, respectively—show an average increase in calorific value of 48·6 per cent on the raw basis, or 16·5 per cent on the dry basis. Their calorific values average as follows: raw fuel, 4970 calories (8950 B.Th.U.), dried fuel, 6340 calories (11400 B.Th.U.), carbonized residue, 7380 calories (13280 B.Th.U.).

(*d*) Nos. 1824 and 984—from Lethbridge and Trochu, Alberta—show an average increase in calorific value of 30·0 per cent on the raw basis and 10·7 per cent on the dry basis. Their calorific values average as follows: raw fuel, 5550 calories (9990 B.Th.U.), dried fuel, 6520 calories (11720 B.Th.U.), carbonized residue, 7210 calories (12970 B.Th.U.).

The above conclusions were deduced from very small scale experiments. They indicate relatively the nature of the residue to be expected from the carbonization of any of the coals tested, and the temperatures at which the operation may best be carried out. Besides the increased calorific value of the residue, other factors must be taken into account in deciding whether any coal is suited for carbonization. Among these may be mentioned the yield of by-products, such as gas and tar oils, and the fitness of the carbonized residue for briquetting purposes. It is with these important factors in view that large scale laboratory experiments are now being carried out.

TABLE I

Coal Sample No. 1665, McCormack Mine, Castor, Alta.

Temperature, degrees C.	Yield of residue	Heat in residue as percentage of heat in original fuel	Analysis of residue			
			Calorific value, calories per gram	Ash	Volatile matter	Fixed carbon
.....105°.....	% 100·0	% 100·0	6100	% 5·0	% 40·7	% 54·3
.....280°.....	93·7	98·4	6400	5·0	35·9	59·1
.....300°.....	91·8	98·0	6485	5·1	34·9	60·0
.....350°.....	87·2	96·2	6730	5·3	30·8	63·9
.....400°.....	81·2	93·4	7020	5·6	25·9	68·5
.....450°.....	75·7	91·1	7330	6·1	20·0	73·9
.....500°.....	71·3	88·5	7520	6·5	15·7	77·8
.....550°.....	67·3	85·7	7740	6·9	12·3	80·8
.....600°.....	64·7	83·5	7850	7·4	8·1	84·5
.....620°.....	63·8	82·4	7880	7·5	6·7	85·8
.....650°.....	62·4	80·0	7765	7·7	5·6	86·7
.....700°.....	61·1	76·9	7640	8·1	3·9	88·0
.....740°.....	60·1	74·8	7605	8·2	3·2	88·6

TABLE II

Coal Sample No. 1790, Midland Collieries, Drumheller, Alta.

Temperature, degrees C.	Yield of residue	Heat in residue as percentage of heat in original fuel	Analysis of residue			
			Calorific value, calories per gram	Ash	Volatile matter	Fixed carbon
	%	%		%	%	%
.....105°.....	100·0	100·0	6290	9·4	40·3	50·3
.....255°.....	97·5	100·0	6500	9·2	39·0	51·8
.....300°.....	94·6	9·74	6555	9·6	35·4	55·0
.....350°.....	88·4	93·0	6700	10·4	30·7	58·9
.....400°.....	82·6	89·4	6840	11·1	26·3	62·6
.....450°.....	76·8	85·8	6990	12·0	21·8	66·2
.....500°.....	71·4	80·7	7100	12·8	16·2	71·0
.....550°.....	68·2	78·2	7210	13·4	12·6	74·0
.....600°.....	65·9	76·4	7300	14·0	9·9	76·1
.....620°.....	65·6	74·0	7325	14·2	9·0	76·8
.....650°.....	63·7	73·9	7270	14·6	7·6	78·4
.....685°.....	62·0	73·2	7300	14·9	5·7	79·4
.....700°.....	61·4	71·7	7275	15·0	5·2	79·8
.....750°.....	59·6	68·6	7220	15·4	4·6	80·0

TABLE III

Coal Sample No. 1795, Sinclair Mine, Hanna, Alta.

Temperature, degrees C.	Yield of residue	Heat in residue as percentage of heat in original fuel	Analysis of residue			
			Calorific value, calories per gram	Ash	Volatile matter	Fixed carbon
	%	%		%	%	%
.....105°.....	100·0	100·0	6430	7·8	38·9	53·3
.....240°.....	97·9	100·0	6565	7·9	37·2	54·9
.....300°.....	96·1	100·0	6690	8·0	35·8	56·2
.....350°.....	90·4	95·1	6810	8·3	31·2	60·5
.....400°.....	81·9	88·9	6985	9·1	24·0	66·9
.....450°.....	75·8	84·3	7160	9·8	17·8	72·4
.....500°.....	70·2	80·3	7340	10·5	12·8	76·7
.....550°.....	68·1	78·7	7440	10·8	10·3	78·9
.....600°.....	66·0	76·5	7490	11·2	8·1	80·7
.....650°.....	64·0	73·6	7420	11·6	5·4	83·0
.....700°.....	61·6	70·2	7305	11·9	4·0	84·1
.....750°.....	60·0	67·9	7220	12·3	2·9	84·8

TABLE IV

Coal Sample No. 1824, Canada West Mine, Taber, Alta.

Temperature, degrees C.	Yield of residue	Heat in residue as percentage of heat in original fuel	Analysis of residue			
			Calorific value, calories per gram	Ash	Volatile matter	Fixed carbon
	%	%		%	%	%
.....105°.....	100·0	100·0	6585	8·4	38·8	52·8
.....250°.....	98·2	99·0	6620	8·4	36·6	55·0
.....300°.....	96·0	98·5	6750	8·5	34·5	57·0
.....350°.....	91·4	95·5	6900	8·8	30·2	61·0
.....400°.....	84·2	89·3	7020	9·5	23·7	66·8
.....450°.....	77·8	83·5	7110	10·3	18·0	71·7
.....500°.....	73·5	79·8	7170	11·0	14·1	74·9
.....550°.....	70·6	77·7	7210	11·4	11·4	77·2
.....600°.....	68·5	75·2	7240	11·8	9·5	78·7
.....650°.....	66·5	72·2	7160	12·1	7·0	80·9
.....700°.....	64·2	69·8	7110	12·4	4·2	83·4
.....715°.....	64·0	69·0	7110	12·5	3·8	83·7

TABLE V

Coal Sample No. 1875, Goodlands, Manitoba

Temperature, degrees C.	Yield of residue	Heat in residue as percentage of heat in original fuel	Analysis of residue			
			Calorific value, calories per gram	Ash	Volatile matter	Fixed carbon
	%	%		%	%	%
.....105°.....	100·0	100·0	5860	9·5	43·5	47·0
.....280°.....	90·9	96·5	6200	10·3	37·0	52·7
.....300°.....	89·5	95·7	6280	10·4	35·6	54·0
.....350°.....	84·2	92·4	6420	10·6	31·2	58·2
.....400°.....	78·5	88·5	6640	11·3	26·2	62·5
.....450°.....	73·7	85·5	6820	11·8	21·8	66·4
.....500°.....	69·8	82·4	6990	12·5	17·8	69·7
.....550°.....	66·8	80·5	7100	13·2	14·2	72·6
.....600°.....	64·0	78·4	7140	13·9	11·0	75·1
.....650°.....	61·6	75·4	7090	14·5	8·0	77·5
.....700°.....	60·0	72·6	7000	14·9	6·0	79·1
.....750°.....	59·5	71·2	6980	15·1	5·0	79·9

TABLE VI

Fuel	C.V. of dry fuel, calories per gram.	Yield[1] at 400° C.	C.V. of residue at 400° C.	Percentage increase C.V. at 400° C.	Optimum[2] temperature, degrees C.	Yield[1] at op. temp.	C.V. of residue at op. temp.	Percentage increase C.V. at op. temp.
		%				%		
No. 1663. Alfred peat	5350	55	6840	27·9	600°	42	7090	32·5
No. 1665. McCormack mine, Castor, Alta	6100	81	7020	15·1	620°	64	7880	29·2
No. 973. Saskatchewan Coal, Brick and Power Co., Shand, Sask	6250	80	7120	13·9	570°	66	7530	20·5
No. 1875. Goodlands, Manitoba	5860	79	6640	13·3	600°	64	7140	21·8
No. 1076. Western Dominion Collieries, Taylorton, Sask	6050	79	6650	9·9	560°	66	7050	16·5
No. 1456. Cardiff Collieries, Ltd., Cardiff, Alta	6100	83	6700	9·8	620°	67	7130	16·9
No. 1505. G. W. Coal Co's. Black Diamond mine, Clover Bar, Alta	6390	83	6970	9·1	600°	67	7490	17·2
No. 1790. Midland Collieries, Drumheller, Alta	6290	82	6840	8·7	620°	65	7320	16·4
No. 1795. Sinclair mine, Hanna, Alta	6430	82	6985	8·6	590°	68	7490	16·5
No. 1507. Tofield Coal Co's. mine, Tofield, Alta	6480	84	6960	7·4	600°	67	7470	15·3
No. 1824. Canada West Coal Co., Taber, Alta	6590	84	7020	6·4	600°	68	7240	9·9
No. 984. Halbert's mine, Trochu, Alta	6430	84	6730	4·7	630°	71	7170	11·5

[1]Yields indicated are those from dried fuel.
[2]Temperature at which highest calorific value is obtained.

TABLE VII

Fuel	Moisture, per cent	C.V.—B.Th.U. per lb.			Percentage increase C.V. of residue at op. temp.			
		Raw fuel	Dry fuel	Residue at op. temp.	Basis of dry fuel	Basis of raw fuel		
						Due to loss of moisture	Due to carbonizing	Total
No. 1666. Alfred peat	25·0	7220	9630	12760	32·5	33·4	43·4	76·8
No. 1665. McCormack mine, Castor, Alta	26·4	8080	10980	14190	29·2	35·9	39·7	75·6
No. 973. Saskatchewan Coal, Brick and Power Co., Shand, Sask	31·8	7670	11250	13550	20·5	46·7	30·0	76·7
No. 1875. Goodlands, Man	31·7	7190	10530	12850	21·8	46·5	32·3	78·8
No. 1076. Western Dominion Collieries, Taylorton, Sask	31·3	7490	10890	12690	16·5	45·4	24·0	69·4
No. 1456. Cardiff Collieries Ltd., Cardiff, Alta	23·6	8390	10980	12830	16·9	30·9	22·0	52·9
No. 1505. G. W. Coal Co's. Black Diamond mine, Clover Bar, Alta	21·5	9040	11500	13480	17·2	27·2	21·9	49·1
No. 1790. Midland Collieries, Drumheller, Alta	14·8	9650	11310	13180	16·4	17·2	19·4	36·6
No. 1795. Sinclair mine, Hanna, Alta	23·2	8890	11570	13480	16·5	30·2	21·4	51·6
No. 1507. Tofield Coal Co., Tofield, Alta	24·5	8800	11660	13450	15·3	32·5	20·4	52·9
No. 1824. Canada West Coal Co., Taber, Alta	12·2	10420	11860	13030	9·9	14·0	11·3	25·3
No. 984. Halbert's mine, Trochu, Alta	17·3	9580	11570	12910	11·5	20·8	14·0	34·8

TABLE VIII

Available Heat in Residue as Percentage of Available Heat in Original Fuel

Temperature, degrees C.	No. 1666 Alfred, Ont.	No. 1665 Castor, Alta.	No. 973 Shand, Sask.	No. 1875 Goodlands, Man.	No. 1076 Taylorton, Sask.	No. 1456 Cardiff, Alta.	No. 1505 Clover Bar, Alta.	No. 1790 Drumheller, Alta.	No. 1795 Hanna, Alta.	No. 1507 Tofield, Alta.	No. 1824 Taber, Alta.	No. 984 Trochu, Alta.
	%	%	%	%	%	%	%	%	%	%	%	%
.....300°.....	89·0	98·0	100·0	95·7	100·0	100·0	99·3	97·4	100·0	99·6	98·5	96·6
.....350°.....	80·0	96·2	96·6	92·4	95·5	96·0	96·6	93·0	95·1	97·6	95·5	92.3
.....400°.....	69·3	93·4	91·4	88·5	87·1	90·9	92·0	89·4	88·9	91·0	89·3	88·3
.....450°.....	63·2	91·1	86·3	85·5	82·3	86·8	86·5	85·8	84·3	84·6	83·5	86·4
....500°.....	60·4	88·5	83·1	82·4	79·5	83·9	82·3	80·7	80·3	81·0	79·8	82·5
.....550°.....	58·1	85·7	80·4	80·5	78·0	81·6	80·1	78·2	78·7	79·0	77·7	81·6
.....600°.....	55·7	83·5	77·1	78·4	74·4	79·5	78·6	76·4	76·5	76·7	75·2	79·8
.....650°.....	53·2	80·0	74·1	75·4	71·4	77·3	75·7	73·9	73·6	74·7	72·2	77·4
.....700°.....	51·4	76·9	71·8	72·6	69·5	75·3	72·3	71·7	70·2	71·3	69·8	75·3
.....750°.....	47·0	74·4	70·0	71·2	67·0	72·0	69·5	68·6	67·9	68·9	67·0	73·0

II

NOTES ON THE BURNING QUALITY OF KEROSENE OILS FOR ILLUMINATING PURPOSES

P. V. Rosewarne

INTRODUCTION

An investigation of the burning qualities of kerosene oils was undertaken at the laboratory of the Fuels and Fuel Testing Division of the Mines Branch, in order to obtain first hand information concerning the relations between the chemical constituents, the physical characteristics and the burning qualities of the oils. Three samples of commercial kerosene were obtained through local dealers who were handling the product of different refiners, and a series of experiments was begun in order to ascertain the test or tests which would indicate the burning qualities most accurately, and which could be performed with a small sample of the oil in a laboratory without elaborate equipment. At the present time the work is not complete, but a preliminary report is appended.

The analyses in Table I define some of the physical characteristics of the kerosene used in the experiments, and indicate the amount of some of the impurities present. The discussion of the results indicates what interpretations have been placed upon similar characteristics and impurities in other oils, by authorities in the past.

CHEMICAL AND PHYSICAL EXAMINATION

The samples of kerosene obtained gave the results shown in Table I.

TABLE I.

Test	Sample No. 1890	Sample No. 1891	Sample No. 1892
Flash point	134°F. (56·7°C.)	136°F. (57·8°C.)	114°F. (45·6°C.)
Viscosity at 80°F. (26·7° C.)	30·0 Secs.	30·8 Secs.	28·8 Secs.
Iodine value	22·6	12·6	15·4
Specific gravity at 60°F. (15·6° C.)	0·8095	0·7910	0·800
Sulphur content	0·057%	0·025%	0·037%
Colour	Stand. white	Super white	Super white

DISCUSSION OF RESULTS

Bacon and Hamor[1] claim that the best oil for burning has the following characteristics (presumably in this order of desirability, although they do not say so definitely):—

1. Satisfactory flash point.
2. Lowest viscosity.
3. Lowest iodine absorption value.
4. Highest specific gravity.
5. Lowest sulphur content.
6. Best colour..

NOTE.—This investigation was undertaken immediately after having received several small samples of kerosene from the Tidal and Current Survey, Department of the Naval Service, with the request that the samples be analysed in order to determine as far as possible the one least likely to smoke when burned in a lamp.

[1] Bacon and Hamor, "American Petroleum Industry," page 876.

Flash Point.—The flash point was determined in the standard Pensky-Martens, closed test apparatus, and in all cases was considerably above the minimum set for kerosenes which may be sold in Canada,[2] namely, 85° F. (29·4° C.). An oil flashing at the minimum temperature would be rated as having a satisfactory flash point. The regulations specify that the flash point be determined in the Abel tester, but the customary practice was followed by first determining the flash point in the Pensky-Martens tester, and only using the Abel tester when the temperature observed was close to the minimum. This method of procedure is followed because the Abel tester is not suitable for a rapid determination unless the flash point is less than 100° F. (37·8° C.).

A low flash point indicates the presence of too large a proportion of the more volatile hydrocarbons, and these would make the oil unsafe to use because of the possibility of the vapours being ignited in the lamp bowl and causing an explosion. Holde[3] draws attention to the statement of Engler, that explosive mixtures form in the lamp reservoir at a temperature about 8° C. higher than that of the flash point observed in the Abel tester. There is, however, very little likelihood of obtaining to-day an oil with too low a flash point, as the lighter fractions have been removed, since they are more valuable than the heavier ones.

Viscosity.—The viscosity was determined in the standard Redwood viscometer.[4] From the results it was found that No. 1892 had the lowest viscosity and No. 1891 the highest. According to Redwood[5] the viscosity of an oil affords, in some cases, a useful indication of its burning quality, but he omits to say in what way this operates. Holde[6] states that the viscosity of a kerosene opposes the rise of the oil in the wick by increasing the resistance in the capillaries. This statement supplies a reason for Bacon and Hamor's claim that the best illuminating oil has the lowest viscosity.

Iodine Value.—The iodine value, which is a comparative measure of the amount of unsaturated hydrocarbons present, was determined according to the Hanus method for gasoline.[7] The lowest iodine value was given by sample No. 1891, and the highest by No. 1890. Holde[8] refers to the experiments of Weger, who claims that the presence of unsaturated hydrocarbons of the olefine, benzene, and partially hydrated cyclic series cause a red coloration of the flame, and thus the flame appears darker. Bacon and Hamor,[9] in referring to the Edeleanu process of refining oil, state that the sulphur dioxide "dissolves unsaturated hydrocarbons and certain aromatic hydrocarbons which are responsible for the unsatisfactory burning and odour of certain kerosenes." It would appear, therefore, that the presence of unsaturated hydrocarbons in an illuminating oil interferes with the proper burning of the oil.

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[2] Redwood, "A Treatise on Petroleum", Vol. III, page 79.
McGill, Bull. 362, Dept. Inland Revenue of Can. "Gasoline", page 4.
[3] Holde-Mueller, "Examination of Hydrocarbon Oils", page 58.
[4] Redwood, "A Treatise on Petroleum", Vol. II, page 275.
[5] Redwood, "A Treatise on Petroleum", Vol. II, page 212.
[6] Holde-Mueller, "Examination of Hydrocarbon Oils", page 55.
Ubbelohde, Petrol., 4, 861 (1909).
Engler, Lewin and Stepanoff, Grundlagen der Lampentheorie, 1906.
[7] Ellis and Meigs, "Gasoline and other Motor Fuels", page 41.
[8] Holde-Mueller, "Examanition of Hydrocarbon Oils", page 65.
[9] Bacon and Hamor, "American Petroleum Industry", page 594.

Specific Gravity.—The specific gravity was determined by the hydrometer method. Sample No. 1891 had the lowest, and No. 1890 the highest specific gravity. The influence of different specific gravities is negligible[10] when considering the amount of oil that can be raised by the wick. A high specific gravity indicates the possible presence of a large percentage of heavy hydrocarbons,[11] or of aromatic hydrocarbons.[12] Redwood[11] states that an undue proportion of heavy hydrocarbons has an injurious effect on the burning quality of an oil. It would seem, therefore, since it has been shown that aromatic hydrocarbons and also heavy hydrocarbons are undesirable in an illuminating oil, and since these two classes tend to increase the specific gravity, that Bacon and Hamor are in error in stating that the best burning oil has the highest specific gravity.

Sulphur.—The sulphur content was determined by the following modification of the combustion method[13]: About 1·3 grams of the sample was burned in an atmosphere of oxygen at a pressure of 500 pounds per square inch in an Emerson calorimeter bomb, in the presence of about 10 cubic centimeters of water. After the oil was burned, the interior of the bomb was rinsed out with water, the washings acidified with hydrochloric acid, the sulphate precipitated as barium sulphate, and the percentage of sulphur calculated. The sulphur content of the oils is shown in Table II.

TABLE II.

Test	Sample No. 1890	Sample No. 1891	Sample No. 1892
	%S	%S	%S
Test 1	0·059	0·029	0·031
Test 2	0·055	0·020	0·040
Average	0·057	0·025	0·037

Much has been written concerning sulphur in petroleum and kerosene oils. Some authorities[14] point out the excessive cost of purifying a crude petroleum having a high percentage of sulphur, some,[15] the vile odour of sulphur compounds such as are found in petroleum, while others[16] dwell on the chemical constitution of the various compounds containing sulphur.

It is universally recognized that a petroleum to be valuable must have a low sulphur content. There is, however, a very confusing lack of agreement among authorities concerning the relations between the sulphur

[10] Holde-Mueller, "Examination of Hydrocarbon Oils", page 55.
[11] Redwood, "A Treatise on Petroleum", Vol. II, page 216.
[12] Campbell, "Petroleum Refining", page 183.
Bacon and Hamor, "American Petroleum Industry", page 595.
[13] Redwood, "A Treatise on Petroleum", Vol. II, page 319.
Hicks, "Mineral Oil Testing", page 60.
Holde-Mueller, "Examination of Hydrocarbon Oils", page 41.
Guttentag, "Petrol and Petroleum Spirits", page 90.
Campbell, "Petroleum Refining", page 49.
Bacon and Hamor, "American Petroleum Industry", page 128.
Cross, "Handbook of Petroleum, Asphalt and Natural Gas", page 332.
[14] Bacon and Hamor, "American Petroleum Industry", page 609.
Redwood, "A Treatise on Petroleum", Vol. II, page 48.
[15] Bacon and Hamor, "American Petroleum Industry", page 609.
Southcombe, "Chemistry of the Oil Industry," page 46.
Cross, "Handbook on Petroleum, Asphalt and Natural Gas", page 136.
Holde-Mueller, "Examination of Hydrocarbon Oils," page 62.
[16] Redwood, "A Treatise on Petroleum", Vol. I, page 239.
Southcombe, "Chemistry of the Oil Industry", page 46.

content of an illuminating oil and the burning qualities of that oil. The results of burning kerosene oils which contain sulphur were collected from various sources, and are summarized as follows:—

Sulphur compounds in kerosene oils: emit an offensive odour;[15] have a deleterious effect on plants, hangings and bindings, when the kerosene is burned in a room in which they are present;[17] cause unsatisfactory burning;[18] cause a white deposit on the glass chimney of the lamp;[19] do not affect illuminating power of the oil unless a derivative of sulphuric acid;[20] cause carbonization of the wick.[21]

No offensive odour was noticeable from any of the three samples of oils tested. The effect on plants, hangings and bindings would be proportional to the amount of sulphur present. Bacon and Hamor do not say in what way sulphur causes unsatisfactory burning. The white deposit on the chimney seems to be a definite result, directly due to sulphur, which can be easily observed. Campbell[22] claims that the removal of all but a trace of sulphurous bodies causes a marked reduction in the amount of the deposit. No other authority available makes any reference to this filming on the chimney. The incrustations on the wick are apparently composed of carbon and are often referred to as toadstools. According to Holde they are due to the presence of esters of sulphuric acid in the oil.

Since the determination of sulphur by the method outlined above includes all the sulphur present, irrespective of the form of combination in which it occurs, it is evident that a small percentage of sulphur so obtained, indicates an illuminating oil more free from the defects just mentioned than an oil having a larger percentage of sulphur. However, the determination of total sulphur cannot be used to distinguish which particular defect, or defects, will be present when the oil is burned.

Colour.—The colour of the samples was determined in Wilson's chromometer. Colour tests for illuminating oils have lost their former importance because various oils on the market to-day are dark-coloured, and nevertheless are quite satisfactory. Exposure to sunlight causes kerosenes to become darker in colour, but does not affect their burning qualities.[23]

EXPERIMENTAL

Apparatus.—The apparatus used consisted of a simple optical bench and shadow photometer; a lamp with an ordinary hinge burner using woven cotton wick, five-eighths of an inch in width; ordinary glass lamp chimneys to fit the above burner; an amyl acetate lamp; a five candlepower, carbon, incandescent, electric light bulb and receptacle; a voltmeter; and a sling psychrometer.

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[15] Bacon and Hamor, "American Petroleum Industry", page 609.
Southcombe, "Chemistry of the Oil Industry", page 46.
Cross, "Handbook of Petroleum, Asphalt and Natural Gas", page 136.
Holde-Mueller, "Examination of Hydrocarbon Oils," page 62.
[17] Redwood, "A Treatise on Petroleum", Vol. II, page 318.
Gill, "Oil Analysis", page 20.
[18] Bacon and Hamor, "American Petroleum Industry", page 609.
[19] Campbell, "Petroleum Refining", page 84.
[20] Holde-Mueller, "Examination of Hydrocarbon Oils", page 64.
Graefe, Petrol, 1, 606 (1905).
[21] Holde-Mueller, "Examination of Hydrocarbon Oils", page 64.
Heussler and Dennstedt, Zeitschrift fur Angewandte Chemie, 17, 264 (1904).
Campbell, "Petroleum Refining", page 181.
[22] Campbell, "Petroleum Refining", page 84.
[23] Holde-Mueller, "Examination of Hydrocarbon Oils", page 52.

The optical bench was merely a planed board, 6 feet long and 8 inches wide, and marked off in divisions of one centimeter. The shadow photometer consisted of an iron rod, and a piece of board, 2 feet long and 8 inches wide, covered with a large piece of white paper for a screen. The amyl acetate lamp was constructed of metal with a cylindrical bowl somewhat after the design of a Hefner lamp. It was mounted on a block of wood of sufficient height to bring the flame to approximately the same level as that of the kerosene lamp. The electric bulb and its receptacle were also mounted on a block of wood of about the same height.

Burning Qualities.—The meaning of the rather vague term, burning qualities, may be made more clear by stating that a good illuminating oil should have, as a rule, the following qualities:—

(a) Minimum smokiness of flame.
(b) Minimum tendency to form incrustations on the wick.
(c) Minimum tendency to form depositions on the chimney.
(d) Low rate of change of illuminating power.
(e) High illuminating power.

Smokiness.—Campbell[24] states that certain oils are very easily affected by slight irregularities in the supply of air to the flame, and smoking of the chimney follows. Continuing, he states[25] that the presence of aromatic and heavy hydrocarbons causes smoking, since after the removal of these constituents by the Edeleanu process the oil does not smoke. Later, he states that the liquid sulphur dioxide used in the process mentioned above also dissolves certain sulphur containing constituents. These two statements make it uncertain whether aromatic hydrocarbons or sulphur compounds cause an oil to smoke. Bacon and Hamor[26] state that liquid sulphur dioxide "dissolves the unsaturated and certain aromatic hydrocarbons, which are responsible for the unsatisfactory burning and odour characteristic of certain kerosenes." The same authorities[27] state that sulphur causes "unsatisfactory burning." If, therefore, an oil burns unsatisfactorily because it smokes, then it would seem reasonable to expect that smokiness is caused by the presence of either sulphur compounds or unsaturated aliphatic hydrocarbons, or aromatic hydrocarbons, or a combination of two or more of these factors.

A method to determine the tendency of an oil to smoke when burned in an ordinary lamp was tried, but it was found to be unsatisfactory with the oils used. Further investigations are to be made along this line.

Burning Tests.—Campbell [28] claims that the only practical test of kerosene is to burn the oil in a lamp, similar to the one in which it is to be used, and to note the diminution of the flame and the incrustation on the wick. An ordinary oil lamp with a large bowl was obtained, and portions of the three samples of kerosene were burned in this lamp. The time for each test was approximately 7 hours. The appearance of the wick and of the glass chimney was noted at the end of the test. Duplicate tests were made, and the results are shown in Table III. From these results it is evident that sample No. 1891 is most satisfactory for burning; that No. 1890 is least satisfactory; and that No. 1892 is intermediate in value.

[24] Campbell, "Petroleum Refining", page 85.
[25] Campbell, "Petroleum Refining", page 180 et seq.
[26] Bacon and Hamor, "American Petroleum Industry", page 594.
[27] Bacon and Hamor, "American Petroleum Industry", page 609.
[28] Campbell, "Petroleum Refining", page 84.
Bacon and Hamor, "American Petroleum Industry", page 876.

TABLE III

Test	Sample No. 1890	Sample No. 1891	Sample No. 1892
Chimney discoloured—			
Test 1	Yes	Very slightly	Slightly
Test 2	Yes	Very slightly	Slightly
Wick incrusted—			
Test 1	Yes	No	Slightly
Test 2	Yes	No	Slightly
Time—			
Test 1	7·0 hrs.	7·0 hrs.	7·5 hrs.
Test 2	7·0 hrs.	5·9 hrs.	7·1 hrs.
Total oil burned—			
Test 1	169 grms.	169 grms.	170 grms.
Test 2	160 grms.	137 grms.	181 grms.
Oil burned per hour—			
Test 1	24·2 grms.	24·2 grms.	22·7 grms.
Test 2	22·9 grms.	23·2 grms.	25·5 grms.
Average	23·6 grms.	23·7 grms.	24·1 grms.

Change in Illuminating Power.—Preliminary experiments were conducted in order to measure the change in illuminating power. Although corrections were made for change of voltage of the electric current supplied to the standard lamp, and for change due to barometric pressure and humidity of the atmosphere, it was found that the use of more elaborate equipment was necessary in order to secure concordant results.

Illuminating Power.—Since unsatisfactory results have been obtained in determining the change in illuminating power, it was decided to postpone a determination of illuminating power until more precise apparatus was available.

COMPARISON OF RESULTS

By combining Table I and Table III, the results of the physical and chemical examination, and the results of the burning tests, are brought together for comparison in Table IV.

TABLE IV

Test	Sample No. 1890	Sample No. 1891	Sample No. 1892
Smoke	None	None	None
Chimney discoloured	Yes	Very slightly	Slightly
Wick incrusted	Yes	No	Slightly
Oil burned per hour	23·6 grms.	23·7 grms.	24·1 grms.
Flash point	134°F. (56·7°C.)	136°F. (57·8°C.)	114°F. (45·6° C.)
Viscosity at 80°F. (26·7°C.)	30·0 Secs.	30·8 Secs.	28·8 Secs.
Iodine value	22·6	12·6	15·4
Specific gravity at 60°F. (15·6°C.)	0·8095	0·7910	0·800
Sulphur content	0·057%	0·025%	0·037%
Colour	Stand. white	Super. white	Super. white

Since sample 1891, when burned, deposits the thinnest film on the chimney and the least incrustation on the wick, it may be said (in the absence of definite results as to tendency to smoke and rate of change of illuminating power), to have the best burning qualities.

The flash point is apparently not indicative of the burning qualities of these oils, since the flash point of No. 1890 and of No. 1891 is much higher than that of No. 1892, although it is apparent that No. 1892 has better burning qualities than No. 1890.

Although there seems to be a distinct interrelation between viscosity, oil burned per hour, and change of illuminating power, the experiments were too few to establish the fact clearly. Further experiments are to be carried out to supplement those already performed.

The iodine value, on the other hand, seems significant. It was found that No. 1891 had the lowest iodine value, that No. 1890 had the highest, and that No. 1892 had an intermediate value. Thus, since actual burning tests showed that No. 1891 was the best oil and that No. 1890 was the worst, it is evident that in this case the lowest iodine value indicates the best oil.

A comparison of the remaining results in Table IV also shows that the lowest specific gravity and the lowest sulphur content indicates the best oil, for the reason given for the iodine value. It is interesting to note that the results of the experiments agree in one respect with the conclusion arrived at theoretically, namely, that the best burning oil has the lowest specific gravity. If further work on a large number of different illuminating oils substantiates the preliminary results obtained from the three oils used, it will then be possible to determine the value of an oil for burning purposes by means of chemical and physical analysis.

CONCLUSION

Some of the burning qualities of three different samples of oil have been determined by means of an actual burning test and the results compared with the results of a chemical and physical examination.

The oil showing the most satisfactory result in the burning test had the following chemical and physical characteristics:—

(a) The lowest iodine value.
(b) The lowest specific gravity.
(c) The lowest sulphur content.

The investigation is to be continued in order to include a larger number of different illuminating oils, and in order to obtain more precise information regarding the illuminating power and the effect upon it of various external conditions.

III

THE LUBRICATING VALUE OF COD LIVER OIL

P. V. Rosewarne

INTRODUCTION

An investigation of the lubricating value of cod liver oil was undertaken in the Division of Fuels and Fuel Testing following an enquiry of the Air Board of Canada for available information respecting "the possible use of cod liver oil as a lubricant in aero engines and a comparison of the lubricating properties of cod liver oil with say pharmaceutical castor oil."

Though the results of the investigation are not positive so far as the

lubricating value of cod liver oil is concerned, nevertheless in view of the use of castor oil as a lubricant, and the apparent paucity of literature upon the subject, the laboratory tests on both these oils appear to be of sufficient interest to justify the following brief record of the investigation.

A summary of the constituents and general characteristics of the two oils, as stated in the literature, may be of value.

CHARACTERISTICS OF CASTOR AND COD LIVER OIL

Castor oil is a vegetable oil expressed from the seeds of the castor oil plant. The oil contains the glycerides of palmitic and ricinoleic acids, as well as those of several other fatty acids.[1] Ricinoleic acid is peculiar to castor oil. Castor oil is classed as a non-drying oil;[2] it solidifies at from −10°C. to −18°C.; its saponification value is not less than 177, nor more than 187;[3] its iodine value is 83 to 90; it decomposes at 300°C.; it is only partially soluble in petroleum distillates.[4]

Cod liver oil is a marine animal oil obtained from the livers of the cod, and other fish. The oil contains the glycerides of palmitic, stearic, clupanodonic[5] and other more complex acids. Clupanodonic acid is characteristic of all fish oils. Cod liver oil solidifies at from 0°C. to −10°C.; its saponification value is not less than 179, nor more than 192; its iodine value is from 135 to 168;[6] stearine separates out on standing.[7]

REQUIREMENTS FOR A GOOD LUBRICANT

Before discussing the results it is to be noted that a good lubricant should possess the following characteristics:—

(a) Sufficient viscosity, or "body," to keep the moving surfaces apart under the maximum pressure.
(b) Chemical inertness, in respect to oxidation and corrosive action on the metals of the bearings.
(c) A low coefficient of friction.
(d) Greatest possible capacity for carrying away heat, so that the bearings may remain cool.
(e) A high temperature of decomposition.
(f) A low solidifying point.

It is unlikely, however, that any one oil or grade of oil will be found to give the best results in every particular. In choosing an oil for a particular machine it is necessary therefore to obtain a lubricant that possesses the quality or qualities which enables the machine to function properly under operating conditions. For that, the two characteristics first mentioned in the above list are fundamental. The remaining four will vary in importance according to the operating conditions. For instance, it is conceivable that an oil having a slightly higher coefficient of friction and a lower solidifying point is to be preferred in some cases, to another oil having a lower coefficient of friction and a higher solidifying point.

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[1] J. B. Cohen, "Theoretical Organic Chemistry".
[2] Holde-Mueller, "Examination of Hydrocarbon Oils", page 360.
[3] British Pharmacopœia, 1914.
[4] Lewkowitsch, "Analysis of Oils, Fats and Waxes", Vol. II, page 399.
[5] Lewkowitsch, "Analysis of Oils, Fats, and Waxes", Vol. I, page 210.
[6] Holde-Mueller, "Examination of Hydrocarbon Oils", page 370.
[7] Laucks, "Commercial Oils", page 78.

RESULTS OF TESTS

The castor oil for the tests was of two grades, (a) commercial oil of good quality, (b) pharmaceutical oil of best quality. The cod liver oil used was of the grade known as "Norway Cod Liver Oil." The laboratory tests gave the results shown in Table I.

TABLE I

Test	Castor oil		Cod liver oil
	(a) grade	(b) grade	
Viscosity, Redwood at 100°F. (37·8°C.)	980 Secs.	1068 Secs.	131·5 Secs.
125°F. (51·7°C.)	460 "	522 "	90·0 "
150°F. (65·6°C.)	242 "	285 "	68·4 "
175°F. (79·4°C.)	146·4 "	153·4 "	56·1 "
210°F. (98·9°C.)	85·7 "	88·0 "	46·5 "
250°F. (121·1°C.)	56·2 "	57·2 "	.....
300°F. (148·9°C.)	43·0 "	44·0 "	35·9 "
Friction test by Thurston oil tester—			
Pressure per sq. inch	.....	40 lbs.	40 lbs.
Speed per minute	.....	814 ft.	896 ft.
Maximum temperature	.....	257°F. (125°C.)*	194°F. (90°C.)
Volume of oil used	.....	1 (approx.)	5 (approx.)
Coefficient of friction at 194°F. (90°C.)	.....	0·05	0·02
Corrosion test on—			
(a) soft brass	Corroded	Corroded	Discoloured
(b) cold rolled steel	Discoloured	.....	No effect
Gumming test	Very slowly	Very slowly	Gums quickly
Flash point (Pensky-Martens, closed test)	446°F. (230°C.)	450°F. (232°C.)	386°F. (197°C.)
Total acidity	6·1%	1·7%	1·1%
Free mineral acids	None	None	None
Saponification value	184	185	191
Iodine value	90	85	154

*The machine was stopped, in order to avoid injury to the bearing, when the thermometer read 257°F.; the temperature was still rising.

DISCUSSION OF RESULTS

Viscosity.—The viscosity of an oil is a measure of the degree of its fluidity, and is closely related to its internal friction. This close relation between viscosity and internal friction, usually expressed as the coefficient of friction, has given rise to a theory that has been very widely accepted as true, namely, that for all speeds and pressures, except for very low speeds and very high pressures, the coefficient of friction is proportional to the viscosity.[8] Battle[9] states that viscosity is also a measure of the combined effects of cohesion and adhesion; that cohesion helps to maintain the film by holding together the particles forming the oil; and that adhesion helps to maintain the film by clinging persistently to the surfaces to be lubricated. Since oils are used for lubrication at widely varying temperatures, the relation between viscosity and temperature is important. If an oil is intended for use at high temperatures, a large variation of viscosity with a change of temperature is undesirable, for the following reasons:—

(a) At ordinary temperatures the oil is so thick that it is inconvenient to handle, in order that it may be of sufficient body at working temperatures.

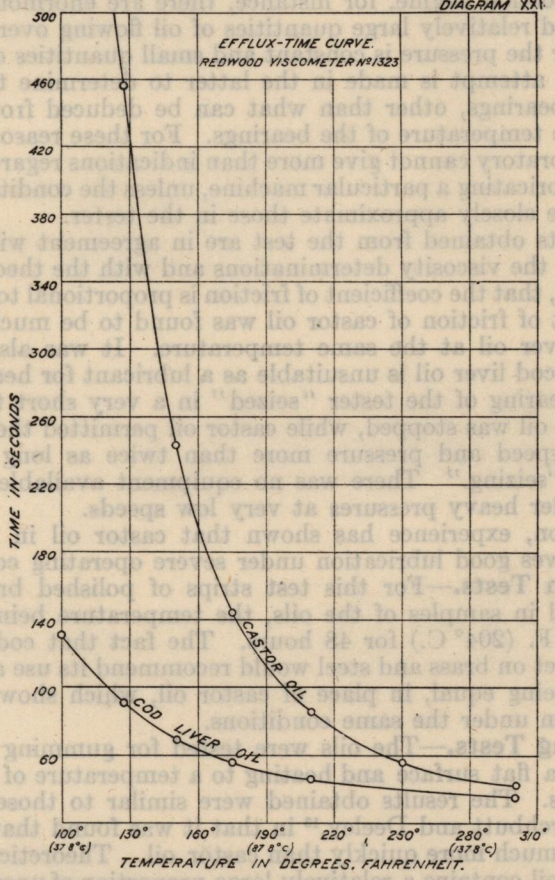
[8] Archbutt and Deeley, "Lubrication and Lubricants", page 84.
[9] Battle, "Lubricating Engineer's Handbook", page 24.
[10] Archbutt and Deeley, "Lubrication and Lubricants", pages 59 and 77.

(b) The pipes and oil ducts of a lubricating system must be large enough to handle the viscous lubricant at ordinary temperatures.

(c) The coefficient of friction tends to be high[10] when starting up a cold machine, and consequently the machine requires a high starting torque.

Accordingly, a lubricant for any machine should possess a viscosity high enough to maintain a lubricating film satisfactorily under the maximum pressure developed, but it should not have any excess viscosity, because that would increase the internal friction, and, if the machine operates at a high temperature, the viscosity should not change greatly with a change of temperature.

DIAGRAM XXI.



The high viscosity of castor oil would therefore indicate its value as a lubricant for heavy machines working at ordinary temperatures. On the other hand, the low viscosity of cod liver oil would seem to make it suitable for light or medium weight machines. In common with many of the more viscous oils the viscosity of castor oil varies greatly with a

change of temperature,[11] while cod liver oil changes comparatively little. The viscosities found are plotted in Diagram XXI, and the curves show graphically this difference in viscosity with change of temperature.

Frictional Tests.—Because of the general application of the theory that the coefficient of friction of an oil is proportional to its viscosity, and because the viscosity is more easily determined than the coefficient of friction, the viscosity is usually taken as an index to the coefficient of friction. However, an actual test of the lubricating properties is often of value, and accordingly castor oil and cod liver oil were tested in the laboratory on a Thurston oil tester.[12]

In discussing a frictional test it must be remembered that an oil tester cannot duplicate the conditions found in all machines. In an internal combustion engine, for instance, there are enormous fluctuations of pressure and relatively large quantities of oil flowing over the bearings; in an oil tester the pressure is constant and small quantities of oil are used. Then, too, no attempt is made in the latter to determine the amount of wear on the bearings, other than what can be deduced from the rate of increase of the temperature of the bearings. For these reasons a frictional test in the laboratory cannot give more than indications regarding the value of an oil for lubricating a particular machine, unless the conditions obtaining in the machine closely approximate those in the tester.

The results obtained from the test are in agreement with the indications given by the viscosity determinations and with the theory mentioned above, namely, that the coefficient of friction is proportional to the viscosity. The coefficient of friction of castor oil was found to be much higher than that of cod liver oil at the same temperature. It was also shown conclusively that cod liver oil is unsuitable as a lubricant for heavy machines, because the bearing of the tester "seized" in a very short time after the supply of that oil was stopped, while castor oil permitted the tester to run at the same speed and pressure more than twice as long without any indication of "seizing." There was no equipment available for studying lubricants under heavy pressures at very low speeds.

In addition, experience has shown that castor oil in the hands of many users gives good lubrication under severe operating conditions.

Corrosion Tests.—For this test strips of polished brass and steel were immersed in samples of the oils, the temperature being maintained at about 400° F. (204° C.) for 48 hours. The fact that cod liver oil has little or no effect on brass and steel would recommend its use as a lubricant, other things being equal, in place of castor oil, which shows a distinctly corrosive action under the same conditions.

Gumming Tests.—The oils were tested for gumming by placing a few drops on a flat surface and heating to a temperature of 248° F. (120° C.) for 8 hours. The results obtained were similar to those of Holde as reported by Archbutt and Deeley,[13] in that it was found that cod liver oil gums or dries much more quickly than castor oil. Theoretically, the fact that cod liver oil contains a relatively large proportion of unsaturated compounds,[14] as shown by the high iodine value, and is readily oxidized,[15] indicates a decided tendency to gum. This tendency would greatly lower the value of an oil for lubricating purposes, because the oil pipes and ducts

[11] Archbutt and Deeley, "Lubrication and Lubricants", page 192.
[12] Redwood, "Treatise on Petroleum", Vol. II, page 297.
[13] Archbutt and Deeley, "Lubrication and Lubricants", page 305.
[14] Lewkowitsch, "Analysis of Oils, Fats and Waxes", Vol. I, page 60.
[15] Encyclopaedia Britannica, Vol. VI, page 636.

would soon clog, to say nothing of the greatly increased friction on,[16] and possible danger to the bearings when a machine on which it has been used is started after standing idle for some time. The gumming property of cod liver oil as it is sold to-day is so great as to almost in itself preclude the use of that oil as a lubricant. Moreover, it is stated that the greater the amount of resin or gum formed by an oil, the more carbon will be deposited in the cylinder of an internal combustion motor.[17]

A series of experiments are being undertaken by this laboratory in order to obtain further information regarding those constituents of cod liver oil that cause gumming, and to investigate methods for its removal or conversion to non-gumming products.

Flash Point.—The flash point of castor oil was found to be satisfactory in every way. The flash point of cod liver oil was found to be slightly below the figure usually required by specifications for mineral oils to be used in internal combustion engines,[18] namely, 400° F. (204·4° C.), and it would accordingly have a greater tendency to burn off the cylinder walls of the engine during the power stroke of the piston. Presumably it would not burn off enough to allow the metal parts to come in contact except under unusual circumstances. The flash point indicates that there would be no appreciable fire risk in handling or storing cod liver oil.

Acidity.—No free mineral acid was found in either cod liver oil or castor oil. There was considerable difference in the amounts of organic acid found in the two samples of castor oil. For determining the amounts of organic acid present the customary practice was followed, namely, the organic acids were calculated in all cases as if they had the same molecular weight as oleic acid. The acid actually present in castor oil is likely ricinoleic, and that in cod liver oil clupanodonic. The corrosive effect is dependent upon the characteristics of the acid present rather than upon the amount. A small amount of one kind may cause more corrosion, under given conditions, than a large amount of another kind of acid. For this reason acidity determinations are only of value when different samples of the same kind of oil are compared, or when the relative corrosive value of the acids present in the two kinds of oil compared is known. In the case under discussion, castor oil has a larger percentage of acid (calculated as oleic acid) and also has a more corrosive action on brass and steel.

Saponification Value.—The saponification values obtained indicate that the different samples of oil may be considered to be unadulterated, since in every case the value obtained approaches the upper limit specified for that particular oil. The adulterants usually added tend to lower the saponification value.

Iodine Value.—The iodine values found indicate that the samples of pharmaceutical castor oil and cod liver oil are pure according to the standard set by the British Pharmacopœia. As mentioned above, a high iodine

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[16] Aisinman, Jour. Soc. Chem. Ind., 14, 282 (1895).
[17] Gill, "Oil Analysis", page 45.
[18] Department of Militia and Defence of Canada, "Provisional Specifications for Lubricating Oil for Motor Cars". U.S. Bureau of Mines, "Report of Committee on Standardization of Petroleum Specifications".

value indicates a large percentage of unsaturated compounds, and this in turn suggests the possibility of the absorption of oxygen to form gums or resins, which would destroy the lubricating qualities of an oil.

Setting Point.—The solidifying point is slightly lower for castor oil than for cod liver oil, and therefore the advantage would be with the former whenever the oil is to be used as a lubricant at low temperatures.

Solid Matter.—The two samples of castor oil remained clear on standing. It was found, however, that a floculent precipitate was formed in cod liver oil on standing. The cod liver oil was perfectly clear at the beginning of the test. Laucks[19] states that under such conditions stearine separates. The precipitate, which was not analysed to determine its composition, would tend to clog the screens and pipes of the oiling system.

Further investigations on stearine from cod liver oil are being carried out and will be published in due time.

Solubility.—The insolubility of castor oil in petroleum distillates gives it an advantage over other oils, such as cod liver oil and all mineral oils,[20] when it is to be used in an internal combustion motor. This advantage is more pronounced in cold weather, when some of the heavier distillates of the fuel condense in the combustion chamber if the cylinder is not up to proper heat.[21] This condensed fuel runs down the cylinder walls, past the piston rings, and mixes with the lubricating oil. If the latter is miscible in all proportions with the distillate, it is soon diluted so much under favourable conditions, that a considerable change in body is apparent. Obviously, if the two liquids are not miscible such a result is impossible.

SUMMARY AND CONCLUSION

(a) Cod liver oil as a lubricant would be suitable for light weight machines were it not for its excessive gumming properties. Castor oil is a more suitable lubricant than cod liver oil for heavy weight machines, since it can support greater pressures, is more durable, and it does not gum.

(b) Cod liver oil has a low coefficient of friction and permits cool bearings, but requires a large supply of oil to the moving parts. Castor oil has a higher coefficient of friction, but does not require a large supply of oil.

(c) Cod liver oil has little or no corrosive action on brass or steel. Castor oil corrodes brass at high temperatures.

(d) Cod liver oil has a lower flash point than castor oil. It is high enough, however, to be fairly satisfactory.

(e) Cod liver oil solidifies at a higher temperature than castor oil.

(f) Solid matter separates from cod liver oil. Castor oil remains clear.

(g) Cod liver oil is miscible with gasoline. Castor oil is only partially miscible.

(h) Cod liver oil as sold at the present time is not a suitable lubricant for aeroplane engines now in use.

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[19] Laucks, "Commercial Oils", page 78.
[20] Lewkowitsch, "Analysis of Oils, Fats and Waxes", Vol. III, page 64.
[21] U.S. Air Service, "Lubrication for Airplane Engines", page 23.

IV

PRELIMINARY REPORT ON THE INVESTIGATION OF OIL SHALES

A. A. Swinnerton

INTRODUCTION

A special study on Canadian oil shales was begun at the Fuel Testing Laboratory in September, 1919, and the results obtained were included in the Mines Branch Summary Report for that year.

The following report contains, in detail, the results of the examination of various oil shale samples sent in by officers of the Department, and by others, and also a full description of the method employed.

All methods for the extraction of oil in commercial quantities from oil shales depend on heat treatment, and among the factors which may affect the character of the shale oil are the following:—the size and kind of retort used; the size of the charge used; and the pressure and nature of the atmosphere within the retort. The best process to employ is not necessarily the one that gives the maximum yield of oil; but the process that gives the maximum yield of oil most suitable for refining purposes is all important, and it was with this objective that the investigation work in this laboratory on oil shales was planned and is being carried out.

According to the original programme, the investigation called for a study of different methods of examining oil shales for their oil yielding qualities with special attention to the nature of the oils obtained. The programme was therefore considered to consist of three parts or stages; first, a study of the different retorting processes on a laboratory scale in order to obtain the maximum oil yield; second, a preliminary examination of the products; third, a more thorough study of the shale oil obtained.

Among the experiments carried out in the early part of the investigation and belonging to part one were the following: (a), the distillation of a given oil shale sample in iron vessels heated by gas flame as used by Leverin[1] for Canadian oil shales, and by the field men of the United States Geological Survey;[2] (b), the distillation of the oil shale sample with and without steam in a specially designed retort surrounded by electric heating elements; and (c), the distillation of the oil shale sample in an electrically heated lead bath furnace, as was used for the lignite carbonization experiments. The experiments under (a) and (b) proved unsatisfactory, due mainly to uneven heating, but those under (c) gave quite satisfactory results. In this apparatus the much desired temperature control was obtainable on a fairly large sized sample of oil shale, and in a few of the runs sufficient oil and other products were obtained for examination, thus allowing some preliminary work to be carried out according to part two of the investigation. Most of the experimental runs in this lead bath apparatus were made at atmospheric pressure. A few runs were, however, made at reduced pressure, and the use of steam was tried.

Further experiments are being made, under part one of the programme outlined, in which the shale is being treated at various pressures in the presence of gases other than air and steam, such, for instance, as hydrogen

[1] Mines Branch Summary Report, 1909, page 153.
[2] United States Geological Survey, Bulletin 641, page 148.

and hot producer gas. The digestion of the oil shale in heavy, high-boiling, petroleum oil, followed by extraction with kerosene and distillation of the residue, according to the Ryan process, is also being tried. These runs are being made on a scale sufficiently large to allow of the preliminary examination of the products according to part two. In future work, parts one and two will be carried out simultaneously, and when the programme for these two parts has been completed, the third part, namely, the more thorough study of the shale oils obtained by the more promising processes will be undertaken, and the results of the completed investigation will, it is hoped, form the basis of a separate bulletin.

A total of 101 samples of oil shale have been examined at the Fuel Testing Laboratory during this period. Of these, 43 samples were submitted by W. J. Wright of the Geological Survey, 28 from Albert Mines, N.B., and 15 from various outcrops along Frederick Brook, N.B. A summary of the results of the analyses and distillations is given in their Memoir 129,[3] pp. 46, 47. Of the remaining 58 samples, 21 were sent in by S. C. Ells of the Mines Branch, from the Pasquia Hills district, Manitoba and Saskatchewan, and 37 were sent in by private individuals from the following localities:—10 samples from Nova Scotia; 3 from New Brunswick, 3 from Quebec, 7 from Ontario, 5 from Manitoba, 8 from British Columbia, and 1 from the North West Territories.

These samples were examined after the preliminary experiments had been carried out, on the form of the retort, on the method of heating, and on the effect of steam and of reduced pressure on the oil yield. It was accordingly considered that sufficient experience and data had been obtained to warrant undertaking the examination of these samples for their oil yielding qualities. In most cases the samples used were large enough to yield sufficient oil and other products on which to make a preliminary examination.

DISCUSSION OF PROCEDURE

The method used may be called a temperature-controlled destructive distillation process in which the reaction proceeds at a predetermined temperature and rate and at atmospheric pressure. This process was chosen because a method at atmospheric pressure seemed more directly applicable to commercial practice than one where reduced or increased pressures were used, and because the preliminary experiments indicated that this method, from among the many tried, was the most reliable for the examination in the laboratory of oil shale samples for their oil yielding qualities. In the apparatus used check results on a given sample were repeatedly obtained, and, according to the weight-balance records, total yields of products approaching 100 per cent of the sample taken were secured. The yield of oil and some of the physical properties, including distillation ranges of the shale oil obtained, are given in the following pages in tabular form. In this connection the results must be considered as comparative and based on a standard oil shale sample, namely, No. 1517 from Albert Mines, N.B., yielding an average of 13·6 per cent of its weight of crude shale oil (that is, nearly 32 imperial gallons per 2,000 lb. ton).

The moisture and ash content of the oil shale samples were found according to standard methods for coal analysis. The calorific value was

[3] Geology of the Moncton map area, pages 46, 47.

determined by mixing a known quantity of a standardized anthracite coal with a finely ground shale and burning the mixture in an Emerson oxygen bomb calorimeter in the usual way. The nitrogen content of the shales was determined by the Kjeldahl method, and the theoretical yield of ammonium sulphate was calculated from the results obtained. The yield of ammonium sulphate given in the tables was calculated on the assumption that 60 per cent of the nitrogen[4] can be recovered commercially. In the destructive distillation tests of the shales no attempt was made to obtain the maximum amount of ammonium sulphate. The crude oil figures reported herewith include the wet oil emulsion and the aqueous distillate from the shale. The water-free oil is the crude oil dehydrated by settling. The dehydrated or water-free oil yields were obtained by slightly warming and agitating the crude oil and then allowing it to settle overnight in a separatory funnel. After running off the water, the oil left was practically water-free. The figures for specific gravity of the oil are corrected to 60° F. The gas yields obtained by distillation are corrected to 60° F. and 30 inches mercury pressure.

DESCRIPTION OF DISTILLATION METHOD USED

The apparatus used consisted of a ✠ section iron retort, a lead bath electric furnace with temperature control, a tubular iron condenser, a gas

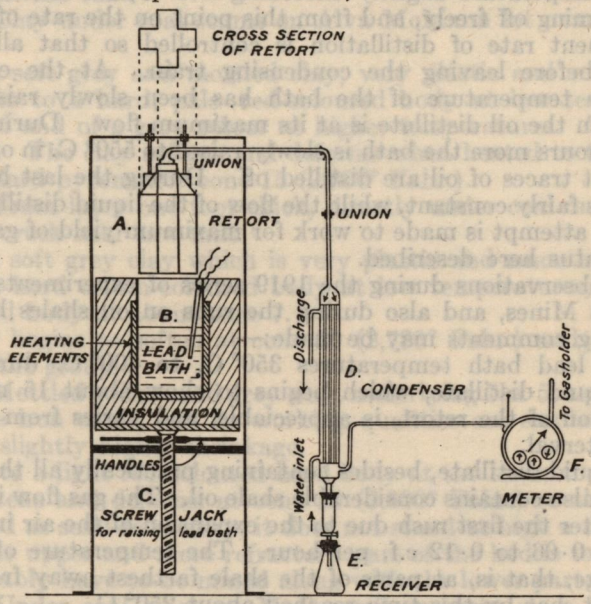

FIG. 12. Apparatus for oil shale distillation.

meter, and gas holders. The capacity of the retort was about 3,500 grams of shale and so designed that no part of the charge was more than 1¼" from the walls of the retort. The temperature lag between the hot lead

[4] R. W. Ells, Joint Report on the Oil Shales of New Brunswick, Nova Scotia and the Oil Shale Industry of Scotland. Part I, page 40. Mines Branch, 1919.

bath and the centre of the charge was thus materially reduced. A special feature of the electrically heated lead bath was that the maximum temperature could be observed at all times by means of a pyrometer, so that the temperature of the bath, and consequently the rate of the decomposition reaction, could be closely followed. A diagrammatic sketch is given in Figure 12, and a more detailed description of the furnace may be found on pages 99–101 of the Summary Report for 1918.

To make a distillation run the following procedure is carried out. The sample is first prepared by crushing to about pea size, a certain amount of fines always being present. The retort A is charged with about 3,300 grams of this prepared shale and the exit pipe is attached by the union at the top of the retort. By means of the four hanging rods the retort is suspended from the top of the angle iron frame and connected with the condenser as in Figure 12. The lead bath B has meanwhile been heated to about 450° C. and is now raised by means of the screwjack C, so that it surrounds the retort. The immersion of the retort cools the lead bath to about 350° C. at which temperature the moisture from the charge comes off freely and the shale is being heated up rapidly, ready for temperature control during the decomposition reaction period.

About 50 to 60 minutes after the immersion of the retort the bath reaches a temperature slightly exceeding 400° C, at which temperature the oil is coming off freely, and from this point on the rate of heating and the consequent rate of distillation is controlled so that all the oil fog disappears before leaving the condensing train. At the end of about 2 hours the temperature of the bath has been slowly raised to about 450° C. when the oil distillate is at its maximum flow. During the course of about 2 hours more the bath is slowly raised to 550° C. in order to insure that the last traces of oil are distilled off. During the last hour the gas rate remains fairly constant, while the flow of the liquid distillate gradually ceases. No attempt is made to work for maximum yield of gas in the lead bath apparatus here described.

From observations during the 1919 series of experiments on oil shale from Albert Mines, and also during the runs on oil shales here reported the following comments may be made:—

(*a*) At lead bath temperatures 350° C. to 400° C. during the first hour the liquid distillate, which begins to show about 15 minutes after the immersion of the retort, is appreciable, and varies from 5–10 cc. per 5 minute interval.

This liquid distillate, besides containing practically all the water from the charge, also contains considerable shale oil. The gas flow is also noticeable, and after the first rush due to the expansion of the air in the system, varies from 0·06 to 0·12 c.f. per hour. The temperature of the interior of the charge, that is, at parts of the shale farthest away from the walls of the retort, has by this time reached about 250° C.

(*b*) At 400° to 450° C. ([5]) bath temperatures, during the second hour, the oil flow increases from about 10 c.c. per 5 minute intervals to as high as 35 to 40 c.c. and remains at this maximum for approximately 15 minutes. The gas rate meanwhile increases to a maximum of about 4 c.f. per hour. It is at this point that the rate of heating is slowed down in order to avoid excessive

[5] Pyrometer temperatures here recorded are subject to a correction of ± 5° C.

tar fog. While the bath temperature is held in the neighbourhood of 450° the maximum rate of flow of both the oil and the gas continues for about a quarter of an hour as mentioned, when the centre of the charge reaches about 400° C. During the next 30 minutes, while the interior temperature does not exceed 425° C., the oil flow decreases to less than 5 c.c. every five minutes. The end of the third quarter of the third hour is now reached. From the beginning of the run the shale during its decomposition reaction which proceeds from the retort walls to the centre of the charge, is at no period subjected to a temperature exceeding 425° to 435° C. The gas rate meanwhile decreases from its maximum flow to about 1 c.f. per hour.

(*c*) During the last 75 minute period of a four hour run, while the bath is raised from 450° to 550° C., the liquid distillate decreases rapidly to less than 1 c.c. every five minutes, and then ceases to flow. The gas rate however remains constant at about 1 c.f. per hour, as above, until the temperature of the centre of the charge at the end of the run is raised to that of the lead bath.

The rates of flow for oil and gas given above apply in general to a shale yielding approximately 30 imperial gallons per 2,000 lbs., and in particular to sample No. 1517 from Albert Mines.

Assuming that the temperature of the inner walls of the iron retort is approximately the same as the lead bath, the difference in temperature between the retort and the charge farthest away from the walls, at the end of the second hour, is about 60°, and it takes approximately one hour from the time the bath reaches, say 400° C., until the centre of the charge has reached the same temperature. The temperature gradient decreases to about 50° at the end of the third hour, when the run is practically finished so far as the oil is concerned.

In view of the following significant observations, viz.: (*a*) that the increase in the oil flow begins as the temperature of the bath reaches 400° C.; (*b*) that this oil flow reaches and maintains its desired maximum until the interior of the charge is raised to about 400° C., and (*c*) that it is apparently necessary to raise the temperature of the interior of the charge only say 25 to 35° C. above 400° C. in order to distil over practically all the oil; it is evident that the temperatures to be employed in order to obtain maximum yields of oil, as the main product from oil shale, are within a fairly narrow range, and that the optimum decomposition temperature is in the close neighbourhood of 400° C.

It is interesting to note that McKee and Ryder[6] in their report of small scale theoretical work on oil shales, check the above observations closely. Their results apparently prove that the decomposition of the kerogen constituents of oil shales takes place within the narrow temperature range of 745° F. (395° C.) and 767° F. (408° C.). These, of course, were temperatures within the retort, and on a very small sample of finely ground shale. It is also interesting to note that their yield of total oil was greater at 425° C. than it was at 410° C. or lower, although this extra yield was only 0·6 per cent of the original shale, and the higher yield contained less heavy bitumen, which they call a first decomposition product, and more light oil, termed by them a secondary decomposition product.

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[6] J. Ind. and Eng. Chem. Vol. 13, No. 7, p. 613 et seq.
J. Ind. and Eng. Chem. Vol. 13, No. 8, p. 678 et seq.

EXPERIMENTAL

In order to determine that the method used in examining the oil shale samples here reported could be checked, three extra runs were made on a sample of Albert Mines shale, which has been reported as run 7, page 29, Mines Branch Summary Report for 1919. This run, No. 7, is reported again as No. 1 in the following table. Runs 2 and 3 were made approximately two years after No. 1, and run 4 about six months later than Nos. 2 and 3. The working sample had meanwhile been kept in a sealed container.

The crude oil figures as used in the weight balance table are significant. In run No. 1, since the water content of the crude wet oil was not checked, the dehydrated oil figure for this run is omitted in the table below.

TABLE I.

Distillation Results of Four Experimental Runs on Sample No. 1517; Oil Shale from Albert Mines

Run No.	Weight balance (p.c. wt. of shale)				Gas and oil yields (per 2,000 lbs.)			
	Crude oil	Gas	Residue	Unaccounted for	Gas		Dehydrated oil	
					Cu. ft.	Density	Sp. Gr.	Imp.g'l.
	p.c.	p.c.	p.c.	p.c.		Air=1		
1	13·4	3·5	82·8	0·3	1,308	0·69	0·852	.....
2	13·6	2·8	82·8	0·8	1,085	0·68	0·870	29·9
3	13·8	3·0	82·4	0·8	1,120	0·71	0·872	30·2
4	13·7	3·5	82·1	0·7	1·040	0·74	0·879	30·2

For reporting the results of the analyses and distillations, the oil shale samples are grouped as follows:—

Group I. Samples marked 1-15 from Tunnel No. 1, Albert Mines, N.B.

Group II. Samples marked 16-18 from Tunnel No. 2, Albert Mines, N.B.

Group III. Samples marked 19-28 from Tunnel No. 3, Albert Mines, N.B.

Group IV. Samples marked 29-47 from Frederick Brook outcrops, Albert Mines, N.B.

Groups I to IV inclusive were sent in by W. J. Wright, of the Geological Survey.

Group V. 21 samples, from Pasquia hills, Saskatchewan district, by S. C. Ells, of the Mines Branch.

Group VI. 37 samples, submitted from various localities, by private individuals.

Figure 13 gives the location of the samples taken by W. J. Wright in the Albert Mines oil shale area, N.B.

Two tables are given for each group, for groups I to IV, the first table reporting the ordinary analyses of the samples and the second table the gas and oil yields by distillation. One table only is given for groups V and VI.

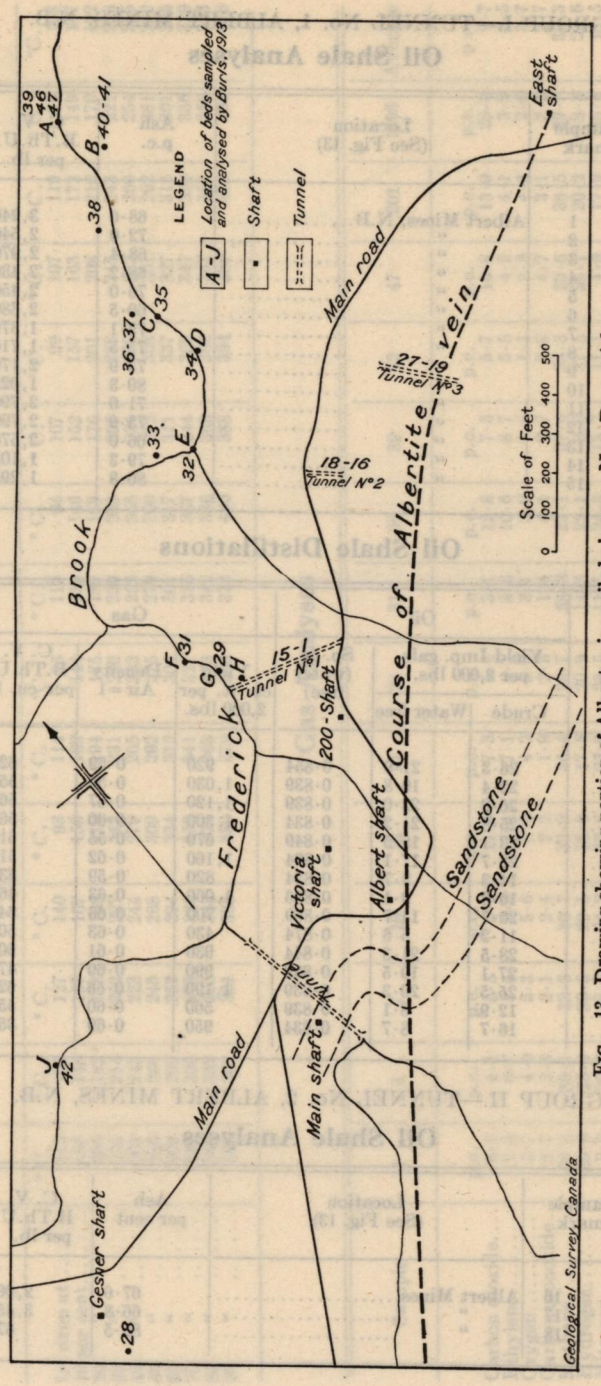

Fig. 13. Drawing showing portion of Albert mines oil shale area, New Brunswick.

GROUP I.—TUNNEL No. 1, ALBERT MINES, N.B.

Oil Shale Analyses

Sample No.	Sample mark	Location (See Fig. 13)	Ash p.c.	C. V. B.Th.U. per lb	Nitrogen p.c.
1518	1	Albert Mines, N.B	68·6	3,240	0·45
1519	2	"	72·0	2,540	0·40
1520	3	"	68·4	2,970	0·44
1521	4	"	68·2	2,430	0·44
1522	5	"	76·0	2,450	0·37
1523	6	"	69·5	2,380	0·43
1524	7	"	79·1	1,370	0·35
1525	8	"	75·5	1,710	0·43
1526	9	"	73·9	2,170	0·42
1527	10	"	80·3	1,025	0·32
1528	11	"	71·0	2,790	0·40
1529	12	"	75·0	2,790	0·53
1530	13	"	66·0	2,570	0·43
1531	14	"	79·3	1,105	0·29
1532	15	"	80·8	1,295	0·50

Oil Shale Distillations

Sample	Oil			Gas			Ammonium sulphate (calc.) lbs. per 2,000 lbs.
	Yield Imp. gals. per 2,000 lbs.		Sp. gr. (water free)	Yield cu. ft. per 2,000 lbs.	Density Air=1	C. V. B.Th.U. per cu. ft.	
	Crude	Water free					
1	26·3	21·9	0·854	920	0·59	620	25·4
2	22·4	19·8	0·839	1,030	0·50	550	22·5
3	26·2	22·0	0·839	1,120	0·67	560	24·8
4	25·8	21·2	0·834	1,300	0·60	560	24·8
5	23·1	18·5	0·849	670	0·55	610	20·9
6	22·7	17·1	0·824	1,160	0·62	510	24·3
7	14·3	8·7	0·824	820	0·59	530	19·8
8	16·3	9·1	0·816	1,060	0·63	460	24·3
9	20·7	14·7	0·869	700	0·66	640	23·7
10	11·2	5·6	0·874	430	0·63	500	18·4
11	28·5	21·3	0·844	930	0·61	600	22·6
12	27·1	19·5	0·839	990	0·69	470	29·9
13	25·5	20·3	0·839	1,190	0·56	620	24·3
14	12·9	8·1	0·839	560	0·60	550	16·4
15	16·7	8·7	0·834	950	0·69	350	28·2

GROUP II.—TUNNEL No. 2, ALBERT MINES, N.B.

Oil Shale Analyses

Sample No.	Sample mark	Location (See Fig. 13)	Ash per cent	C. V. B.Th.U. per lb.	Nitrogen per cent
1533	16	Albert Mines	67·6	2,000	0·36
1534	17	"	66·8	3,450	0·61
1535	18	"	82·5	670	0·33

Oil Shale Distillations

	Oil			Gas			Ammonium sulphate (calc.) lbs. per 2,000 lbs.
	Yield Imp. gals. per 2,000 lbs.		Sp. gr. water free	Yield cu. ft. per 2,000 lbs.	Density Air=1	C. V. B. Th. U per cu. ft.	
	Crude	Water free					
16	20·1	16·5	0·859	980	0·74	510	20·3
17	30·5	23·9	0·839	940	0·60	650	34·4
18	10·6	6·0	0·854	470	0·65	420	18·6

GROUP III.—TUNNEL No. 3, ALBERT MINES, N.B.

Oil Shale Analyses

Sample No.	Sample mark	Location (See Fig. 13)	Ash per cent	C. V. B.Th.U. per pound	Nitrogen per cent
1536	19	Albert Mines	72·2	2,680	0·44
1537	20	"	88·2	90	0·33
1538	21	"	79·1	880	0·26
1539	22	"	68·5	2,930	0·42
1540	23	"	83·3	630	0·47
1541	24	"	86·5	70	0·45
1542	25	"	78·1	1,025	0·41
1543	26	"	75·1	1,710	0·38
1544	27	"	75·5	1,260	0·41
1545	28	(Gesner shaft)	76·0	1,700	0·31

Oil Shale Distillations

Sample	Oil			Gas			Ammonium sulphate (calc.) lbs. per 2,000 pounds
	Yield Imp. gals. per 2,000 lbs.		Sp. gr. water free	Yield cu. ft. per 2.000 lbs.	Density Air=1	C. V. B.Th.U. per cu. ft.	
	Crude	Water free					
19	27·2	19·8	0·854	870	0·64	580	24·8
20	15·4	Trace	.....	240	.....	.....	18·6
21	11·6	7·6	0·857	380	0·71	540	14·7
22	30·0	24·4	0·844	900	0·62	640	23·7
23	11·2	5·6	0·849	550	0·79	400	26·5
24	12·4	Trace	.....	440	0·90	380	25·4
25	14·6	9·2	0·849	770	0·83	280	23·1
26	20·4	14·2	0·839	790	0·44	590	21·4
27	15·2	8·8	0·860	560	0·77	330	23·1
28	18·2	12·4	0·859	640	0·70	460	17·5

GROUP IV.—FREDERICK BROOK OUTCROP, ALBERT MINES, N.B.

Oil Shale Analyses

Sample No.	Sample mark	Location (See Fig. 13)	Ash per cent	C. V. B.Th.U. per pound	Nitrogen per cent
1546	29	See accompanying map	73·0	1,560	0·45
1547	31	"	63·0	3,040	0·44
1548	32	"	76·5	1,495	0·58
1549	33	"	75·4	2,720	0·48
1550	34	"	73·8	1,655	0·49
1551	35	"	65·1	4,105	0·60
1552	36	"	77·3	1,880	0·50
1553	37	"	78·0	1,440	0·25
1554	38	"	69·0	3,460	0·58
1555	39	"	68·0	3,545	0·54
1556	40	"	65·3	4,210	0·66
1557	41	"	81·2	820	0·21
1558	42	"	68·2	4,090	0·71
1587	46	"	75·8	2,230	0·41
1588	47	"	83·5	1,235	0·30

Oil Shale Distillations

Sample	Oil			Gas			Ammonium sulphate (calc.) lbs. per 2,000 lbs.
	Yield Imp. gals. per 2,000 lbs.		Sp. gr. water free	Yield cu. ft. per 2,000 lbs.	Density Air=1	C.V. B. Th. U. per cu. ft.	
	Crude	Water free					
29	18·5	12·7	0·854	700	0·70	440	25·4
31	27·0	23·8	0·844	820	0·62	770	24·8
32	19·7	17·3	0·869	900	0·75	450	32·7
33	19·5	16·7	0·864	930	0·72	480	27·1
34	14·4	13·2	0·864	1,330	0·82	370	27·7
35	22·4	16·6	0·834	1,180	0·43	630	33·9
36	14·5	12·7	0·884	1,060	0·75	480	28·2
37	12·3	11·3	0·864	470	0·83	540	14·1
38	32·0	28·0	0·874	1,485	0·88	580	32·7
39	26·2	25·2	0·864	1,060	0·70	750	30·5
40	27·7	26·5	0·864	1,350	0·72	660	37·8
41	6·0	4·8	0·864	1,180	0·98	720	11·9
42	40·2	32·2	0·859	1,550	0·52	570	40·0
46	17·6	16·6	0·854	1,080	0·92	440	23·1
47	10·7	9·1	0·855	900	0·60	500	16·9

GROUP V.—PASQUIA HILLS DISTRICT, SASKATCHEWAN AND MANITOBA

Oil Shale Distillations

Sample No.	Location	Oil yield Imp. gals. per 2,000 lbs.		Sp. gr. oil Water free	Gas yield cu. ft. per 2,000 lbs.	Ammonium sulphate* lbs. per 2,000 lbs.
		Crude	Water free			
1825	Man River	20·9	5·7	0·954	840	5·4
1827	"	13·4	5·0	0·964	570	2·9
1830	"	15·2	6·1	0·951	890	6·7
1831	"	15·3	5·5	0·946	800	4·8
1826	"	18·0	5·1	0·950	750	8·7
1828	"	18·5	5·8	0·942	700	3·1
1837	"	16·1	9·4	0·954	700	2·2
1838	"	11·9	6·5	0·944	500	2·9
1840	"	17·8	4·0	0·945	620	2·9
1842	"	14·9	Trace	.....	.....	.....
1843	"	15·6	Trace	.....	.....	.....
1850	Carrot river	13·2	6·0	0·959	730	1·7
1851	"	12·4	4·9	0·958	600	2·8
1852	"	10·4	3·2	0·953	410	2·3
1853	Jordan river	24·4	10·9	0·964	1,130	3·0
1858	Cracking river	22·5	10·9	0·957	.....	2·6
1860	"	21·1	9·1	0·957	900	2·8
1861	"	22·1	8·9	0·957	880	2·8
1862	"	.....	7·0	0·966	.....	.....
1863	"	24·4	12·8	0·949	920	4·5
1895	Ochre river	12·5	4·1	0·953	500	1·0

*The yield of $(NH_4)_2 . SO_4$ from these samples was obtained directly by passing the gases through dilute sulphuric acid.

GROUP VI.—MISCELLANEOUS SAMPLES

Oil Shale Analyses and Distillations

Sample No.	Location	Moisture %	Volatile matter %	Ash %	Nitrogen %	C.V. B.Th.U. per lb.	Oil yield Imp. gals. per 2,000 lbs. Water	Oil yield Imp. gals. per 2,000 lbs. free
1572	Abitibi river, Ont.	0·4	26·4	61·4	0·20	420		1·6
1573	" "	1·4	11·7	84·8	0·37	1,180		5·6
1575	" "	1·7	10·2	86·6	0·41	1,070		3·9
1514	Saanich, B.C.	1·2	6·8	88·2	0·19		Trace	
1576	Sydney, N.S.	0·9	44·3	50·6	0·36	3,460		13·5
1579	Indian mountain, N.B.	0·6	0·6	85·5	0·15		Trace	
1633	Pasquia hills, Man.			74·8		2,690		4·0
1637	" "			62·9		1,990		4·0
1517	Albert Mines, N.B.	0·7	31·4	65·8	0·8	4,210		30·1
1707	Fernie, B.C.	1·6	7·1	91·4				
1708	"	1·5	6·6	91·2		240		
1709	"	1·8	9·0	90·6				
1714	"	1·3	14·8	84·3				
1715	"	1·3	14·1	85·5				
1723	New Glasgow, N.S.	1·8	18·2	69·5	0·7	4,000		16·0
1728	Manitoulin island, Ont.	0·3	36·3	63·2	0·15	1,730		4·8
1729	" "	0·5	33·2	66·2	0·20	1,710		8·1
1730	Northern Manitoba	10·1	18·7	67·6			Trace	
1734	British Columbia	1·7	13·2	82·8	0·28			5·0
1735	"	1·7	35·9	63·7	0·30			4·0
1777	Mafekin well, Man.	1·0	33·1	65·7		890		
1778	New Glasgow, N.S.	0·8	33·9	58·4				50·4
1779	" "	0·7	31·5	46·1				29·8
1780	" "	1·0	21·8	62·2				17·9
1781	Westville, N.S.	0·5	31·4	72·0				
1812	Cape Breton, N.S.	1·1	14·2	84·7	0·54	970		
1813	" "	1·0	7·7	92·2	0·37			
1814	" "	0·6	13·6	85·4	0·70			
1815	" "	1·1	12·8	85·7	0·65			
1873	Ericksdale, Man.	3·2	8·9	83·8			Trace	
1894	Cantamount stn., N.B.	1·0	10·9	85·6				2·0
1881	Mackenzie River dist.							7·0
1920	Metagami river, Ont.							10·0
1921	" "							10·0
1913	Gaspe peninsula, P.Q.						Trace	
1918	" "							
1922	" "							24·0

EXAMINATION OF THE OIL AND GAS

A total of 46 distillation tests were made on the oil obtained from the oil shale samples, by the method described in the earlier part of this report. Since samples from the same area gave oils, which on distillation gave quite similar results, the distillation range of every oil is not reported here. In Table II, however, is given the distillation range of fourteen different oils, the first twelve from shale of the Albert Mines, N.B., area, and No. 1895 from Pasquia hills, Saskatchewan. Figure 14 shows the curve plotted from the average of these fourteen shale oils.

The method followed in examining these oils was the same as used for standard gasoline distillations, where 100 c.c. of the sample to be examined are distilled from an Engler flask of standardized dimensions at the rate of about 5 c.c. per minute.

Table II also gives the analyses of the gases from the fourteen different oil shales and their average. Future reports on the investigation of Canadian oil shales will include more details of the properties of the shale oils obtained by various methods of extraction.

TABLE II

Oil Distillation Ranges

Sample	1	4	7	10	16	21	28	31	36	39	42	47	201	1895	Average
	° C.	° C.	° C.	° C.	° C.	° C.	°C .	° C.	° C.	° C.	° C.	° C.	° C.	° C.	°C.
1st drop at.....	112	100	127	140	93	114	95	119	94	107	99	107	119	79	108
10 per cent.....	168	148	157	167	153	162	146	159	146	152	157	163	172	149	157
20 "	200	182	178	198	210	194	175	189	183	176	201	208	210	170	192
30 "	235	214	202	227	242	231	222	220	223	206	238	245	243	221	226
40 "	262	243	223	244	279	265	252	252	268	234	278	282	266	254	255
50 "	286	268	245	268	309	292	275	279	300	272	308	301	292	281	282
60 "	309	289	262	294	334	315	304	301	317	311	337	327	315	301	305
70 "	331	309	273	320	353	341	327	325	357	344	358	347	339	336	329
80 "	348	332	293	359	374	363	363	342	386	363	382	367	357	350	353
90 "	382	362	324	385	389	379	366	373	400	388	394	386	378	361	376

Gas Analyses

Sample	1	4	7	10	16	21	28	31	36	39	42	47	201	1895	Average
	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.
Carbon dioxide....	8·5	4·4	5·5	5·2	11·9	7·3	5·4	3·1	12·8	7·5	5·7	16·8	13·9	28·0	9·7
Ethylene.....	7·3	6·7	5·5	6·0	7·4	7·1	6·1	8·6	10·6	8·7	5·5	4·5	4·1	3·8	6·5
Oxygen.....	2·7	4·0	5·3	5·6	3·9	4·2	6·9	3·1	3·0	1·8	1·1	2·3	4·3	2·6	3·7
Carbon monoxide..	2·7	2·3	2·1	2·4	2·2	1·9	2·5	1·4	4·1	2·7	3·8	4·7	3·1	2·2	2·7
Methane.....	40·3	36·1	32·6	30·7	32·0	36·4	28·7	58·0	23·1	57·6	35·3	29·6	36·5	29·7	36·3
Hydrogen.....	29·6	25·9	30·5	29·5	18·8	17·5	20·9	14·3	22·1	8·0	37·5	38·1	20·2	8·3	22·6
Nitrogen.....	8·9	20·6	18·5	24·2	23·8	25·6	29·5	11·5	24·3	13·7	11·1	4·0	17·9	25·4	18·5
Inflammable gas..	79·9	71·0	70·7	65·0	60·4	62·9	58·2	82·3	59·9	77·0	82·1	76·9	63·9	44·0	68·1
B.Th.U. per cu. ft. (calc.).....	620	560	530	500	510	540	460	770	480	750	570	500	500	390	550

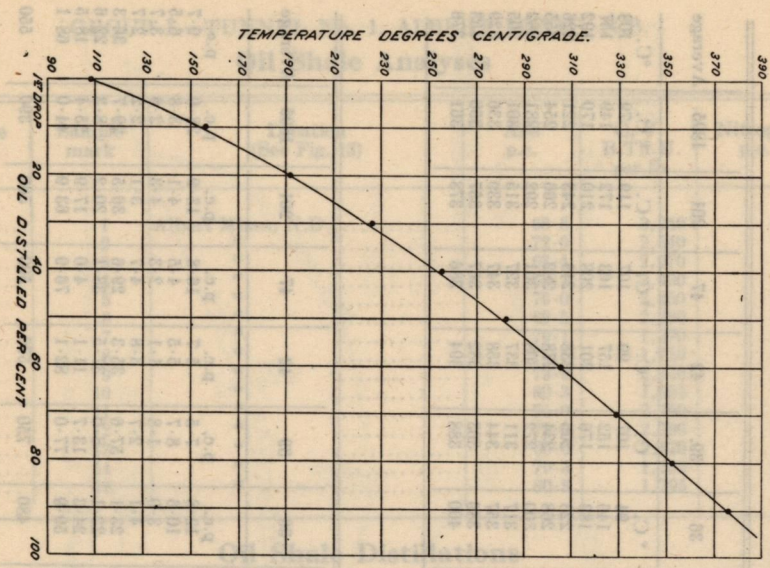

FIG. 14. Average distillation curve of shale oils from Table II.

CERAMIC DIVISION

Joseph Keele
Chief of Division

I

OUTLINE OF WORK DONE

The field and laboratory work of the Ceramic Division for the past season consisted in the investigation of raw materials used in the ceramic industries. These include clays, shales, feldspar, quartz, limestone, talc, bauxite, magnesite, etc.

A large number of samples of clays and shales were tested for people in various parts of the Dominion; but as most of these samples were not very intelligently collected, and the localities from which they came not very clearly indicated, the results of the tests are not recorded.

In view of the fact that most people do not know how to examine and sample a deposit of clay or shale, it has been deemed advisable to devote a part of this summary report to giving definite instructions to unskilled prospectors, and others, relative to the proper way to select samples from promising deposits.

The proper sampling of deposits, especially those liable to variation at depth, or those containing impurities, is very important.

Certain clays which might be utilized for the manufacture of pottery were tested in the laboratory, and the results are given in the following pages. These clays are of low grade, but when prepared are quite plastic and smooth, so that they may be used for the manufacture of ornamental pottery, or for instruction in modelling in schools where training in manual arts is given.

Refractory clays are of rare occurrence in Canada, and no new localities for them were recorded during the past season, but the occurrence of semi-refractory clays from new localities in New Brunswick is recorded in this report.

There are large areas in Canada, especially in the province of Ontario, where stony clays abound, and where stoneless clays occur only in small thin patches which are easily exhausted. Many attempts have been made to work stony clays, but very few are successful. A section of this report gives a description of the methods generally used in overcoming the difficulty.

The white scum which sometimes obscures the colour of the surfaces of brick is objectionable if they are to be used for facing purposes, and brick having this defect are frequently rejected by architects. The method most generally used for curing scumming is described in this report.

The field work for 1920 consisted in gathering data for a bulletin on the clay and shale deposits of Ontario. A journey was also made to the Grand Lake coal area in New Brunswick for the purpose of sampling clays in the coal mines.

II

TESTS OF CLAYS FROM VARIOUS PROVINCES

Nova Scotia

Avonport.—A series of experiments was made for Messrs. L. E. Shaw & Co., Avonport, Nova Scotia, on mixtures of plastic brick clay used at their plant, and shale which outcrops in the vicinity.

The object of the tests was to find out if the shale was a suitable material for making clay products, and if it could be used in combination with the clay for making certain wares for which the clay alone was unsuited.

Although the shale when ground to pass a 10 mesh screen was found to be plastic enough when wet to flow through a stiff mud brick die, it was not plastic enough to pass through a hollow block die without tearing, furthermore the burned colour was not good, and hence brick made from the shale alone could not be used for facing brick.

A mixture of equal parts of clay and ground shale was found to give perfect results in working through a hollow die, or for making rough texture facing brick, as the mixture produced a much better red colour than the shale alone.

It is possible to make hollow ware by using two parts of clay to one of shale, thereby lessening the expense of grinding the shale, but for rough texture face brick, equal parts of clay and shale give the best results.

By adding to their equipment a dry pan for grinding shale, this plant was able to make face brick and fireproofing, whereas hitherto it was restricted to the manufacture of common brick and field drain tile. The temperature of burning the new product was not increased to any appreciable extent over that of burning the ware already made.

St. Margarets Bay.—A sample of so-called kaolin was collected by Mr. E. R. Faribault, of the Geological Survey, from a deposit on the property of Mr. David McLean, at the head of St. Margarets bay, Halifax county.

The sample submitted consisted principally of fragments of granite with some yellowish silty clay, and some small lumps and particles of soft white material. The white lumps are probably the magnesium mineral known as saponite.

The sample when washed through a 200 mesh screen yielded only 20 per cent of clay. This clay when made up into bricklets and burned had an excessive shrinkage and a dark red colour. It softens and deforms at a comparatively low temperature.

The material is of no economic value.

New Brunswick

Grand Lake Coal Area

The Carboniferous rocks in the Grand Lake basin contain a thin coal seam which has been mined at several points for a number of years. In order to provide headroom for mining, about two feet of the shale overlying the coal is removed and brought to the surface, where it is piled in waste dumps.

This shale is suitable for the manufacture of red building brick, and possibly for floor tile, or other vitrified products. A detailed report on the shales of this locality is given in Memoir No. 44, Department of Mines, Ottawa, 1914.

Several new openings on the coal seam have been made in the last few years, and the clays and shales at these points have been sampled by Mr. W. B. Evans, of Rothwell, and forwarded to the Mines Branch for examination.

The clays underlying the coal seam are more refractory than those overlying the coal, hence most of the samples were underclays. The underclays were found to vary in colour in the raw state and in working and burning properties. Some of the samples were smooth and plastic and might be classed as stoneware clays, suitable for making pottery. None of the samples were found to be refractory enough to be classed as fireclays, although they stood enough heat to be called semi-refractory, and are useful clays for many purposes, especially for making vitrified wares. The thickness of the underclays is not very great, and as far as could be ascertained do not exceed four feet, and some of them are less. Some of the beds can be obtained near the surface by stripping the overburden, but others are obtained by underground mining and could probably be taken out with the coal.

No. 4.—Clay under coal seams on Alva McMann lot near Newcastle wharf.

This is a soft grey and mottled clay, very plastic and smooth when wet. It burns to a hard, pale red-coloured body at low temperatures, but is vitrified and of a grey colour at higher temperatures. The shrinkages are high. The clay softens and deforms when heated to the bending point of standard pyrometric cone 15 (2,600° Fahr.).

No. 5.—From layer one foot thick, directly under coal on south side of Newcastle creek near saw mill.

This is a soft grey clay which is very plastic and smooth when wet. It burns to a hard cream-coloured body at low temperatures, and forms a grey vitrified body at about cone 7.

The clay begins to soften at cone 20 (2,786° Fahr.) and is the most refractory of the group.

No. 6.—Mottled yellow and grey clay, underlying No. 5, and evidently contains more iron oxide, but its working qualities are similar to No. 5, and it has a slightly higher shrinkage.

It burns to a light red-coloured hard body at the lower temperatures. This clay softens and deforms at cone 15 (2,600° Fahr.) so that while it may be classed as semi-refractory, it does not stand as high a temperature as No. 5. The upper and more refractory part of the bed is too thin, and it would probably have to be mined along with the lower part.

No. 7.—Under coal seam on McDougall property close to northern bank of Newcastle creek.

Dark grey clay carrying particles of coal. It is very plastic when wet and works very smoothly, but its shrinkage in drying is rather high.

It burns to a porous pink body at low temperatures, but becomes vitrified at about cone 7.

This clay softens and deforms at cone 17, so that it has semi-refractory properties.

No. 8.—Under coal seam on shore of Syphers' cove, Grand lake. Brown, grey and rusty mottled clay, containing much grit.

This clay is not quite so plastic and smooth as the others and its shrinkage is less.

It burns to a pale salmon-coloured, porous body at a low temperature.

This clay softens and deforms at cone 10 (2,426° Fahr.) so that it is not even semi-refractory.

No. 9.—Clay under coal seam exposed in ditch along road on hill at south side of Little river.

Soft yellow shale with fairly good plasticity. It burns to a hard red body at low temperatures. This shale has semi-refractory qualities, as it does not deform below the softening point of cone 17 (2,676° Fahr.).

No. 12.—Clay under coal seam on south bank of Little river, one mile above road.

Soft dark mottled clay, fairly smooth and plastic when wet. This clay gave practically the same results in testing as No. 9.

No. 10.—Hard grey shale, five feet thick, under coal seam on the stripped area of Rothwell property.

This shale was ground to pass a 12 mesh screen, but the ground shale has only a low plasticity when mixed with water, so that it was difficult to mould it into shapes. It burns to a porous red body at low temperatures, but will soften and deform at about cone 5 (2,246° Fahr.).

The working qualities of this shale would be improved by the addition of some of the plastic underclays from another locality, and the mixture could possibly be used for making fireproofing, or paving brick.

Summary of Tests.—The results of the tests made in these clays prove that some of them might be used for foundry clays, or for making stove linings and other products in which a high heat resistance is not demanded. Their plasticity, working and vitrifying qualities are good enough to recommend them for the manufacture of sewer pipe, or electrical conduits, but as their shrinkage is rather high they would require to be mixed with some of the hard shales in the district to remedy this defect.

A similar mixture would also be necessary in producing face brick and fireproofing.

Some of the clays would be suitable for making stoneware household articles, and ornamental pottery.

St. Andrews.—A sample of clay from the bank of the St. Croix river at Bayside, 6 miles from St. Andrews, was tested for Mr. B. Kane, Charlotte County Cottage Crafts.

This is a grey, slightly calcareous, silty clay, apparently free from pebbles, or coarse grit. It developed fairly good plasticity when wet and works easily for moulding. The drying qualities are good and the shrinkage on drying is low, being only 4·5 per cent. It burns to a light red and porous but hard body at cone 010, the total shrinkage at this temperature being 6 per cent and the absorption 13 per cent. When burned to cone 06 the test pieces are darker in colour and the body harder, the total shrinkage being 9 per cent and the absorption 6·5 per cent. When burned to still higher temperatures the shrinkage is greater and the material is liable to soften and deform.

This clay would be suitable for making good hard common red building brick by either the soft mud, or the wire cut process, and as its tensile strength in the raw state is good it could probably be used in the manufacture of field drain tile, but it could not be used for making vitrified ware

Ontario

Kenora.—Six samples of clay and sand were sent by the Keewatin Lumber Company, of Kenora, to the Mines Branch for testing their suitability in the manufacture of brick and drain tile. The materials are glacial lake clays, varying in colour from light grey to black, and the sands are apparently from the raised beaches near the Lake of the Woods.

These clays have fairly good plasticity and working qualities, but their drying qualities are not good. They require the addition of sand in order to reduce the shrinkage and assist in the drying. The clays, however, when mixed with sand are capable of making good red building brick, or field drain tile, but they are not suitable for the manufacture of vitrified ware.

Finmark.—A sample of clay from Finmark was sent for testing by the Division of Development and Resources, Canadian Pacific railway. This was fine-grained reddish calcareous clay, with streaks and layers of grey silt.

This clay is very plastic and has good working qualities, but its shrinkage on drying is 9 per cent, which is rather high.

It burns to a dense hard red body at 1,700° Fahr., but when burned to 1,800° the clay shrinks greatly and is near its softening point.

This clay would probably be suitable for the manufacture of brick and drain tile if about 30 per cent of sand was added to it.

The clay when tested for pottery was found to contain some particles of coarse grit, so that it had to be washed and screened.

The washed clay worked well on the potter's wheel so that any desired shap could be made, but the shrinkage is rather too great. A mixture of washed clay to which 35 per cent of flint was added gave a good working body, and when coated with a white slip inside, and the whole covered with clear glaze, would be suitable for cooking ware.

Desbarats.—A sample of clay from Desbarats lake was submitted for testing by Mr. Jacob Olsen, of Desbarats. This is a reddish calcareous clay, with white films, and layers of silt, very similar to the clay from Finmark.

This clay is very plastic and stiff when wet and is difficult to work. Its working and drying qualities are improved by the addition of 35 per cent of sand. The mixture yields a good hard building brick of light red colour.

Kingston.—Six samples of clay, loam and sand were submitted for testing, by Mr. W. M. Goodwin, M.E., and were collected on the east half of lot 23, in the fourth concession of Kingston township.

The clays from this locality are very stiff and difficult to work when wet. The shapes moulded from them are difficult to dry and the shrinkage in drying is excessive.

Samples of loamy and sandy clays are low in plasticity but would probably be suitable for making common red building brick by the soft mud process.

Mixtures of the stiff clay, the loamy clay, and sand were used, and resulted in very good common red building brick, but on account of the difficulty in drying, they would have to be dried outdoors on racks and pallets, and not indoors in an artificially heated dryer, as fast drying would result in too many cracked bricks.

III

POTTERY CLAYS

The Summary Reports of the Mines Branch for 1919 and 1920 contained chapters on tests of pottery clays from various parts of the Dominion. A short description of how pottery of various kinds is made, and the compositions of glazes and enamels with which they are coated, is included in the 1919 report. Clays from some additional localities were procured, and tested during the past season, the results of which are given in the following pages.

Surface clays, generally of glacial origin, are the principal materials upon which work was done, as these are the only kind available over the greater part of Canada.

The higher grade of clays such as stoneware clays, fireclays and kaolin, are comparatively scarce in this country, and are restricted to a few localities. The principal accessible source of stoneware clay is confined to certain areas of southern Saskatchewan,[1] but a small quantity is also found in Nova Scotia and New Brunswick.

As the glacial clays will not stand a very high fire, the glaze that is applied to them must be easily fusible, otherwise the ware will distort when fired for the second time after the glaze coat is applied.

Nearly all the surface, or glacial clays, contain more or less grit, and are not smooth enough to be shaped on the potter's wheel, consequently, they must undergo preparation which will eliminate the coarse material. This is accomplished either by slaking the clay in water and passing it through a screen, or by drying the clay and pulverizing it, but washing and screening is the method generally employed.

Heavy household pottery, such as butter crocks, mixing bowls and jardinieres, can be made from the washed glacial clays. The only objection to their use is that the body is porous, and when the glaze becomes accidentally chipped off, a spot which gathers dirt results. The stoneware articles are not open to this objection, as even when the glaze chips the vitrified body beneath will not absorb water.

The glacial clays cannot be vitrified, because when the vitrification point of these clays is approached, softening and distortion takes place, consequently the temperature at which the ware is burned is kept well below their vitrification temperature.

While household pottery made from surface clays may not be able to compete with those made from stoneware clay, except in a limited way, there is a large demand for glazed ornamental ware, which has a distinctive treatment both in form, colour and texture, and this demand can be supplied in part by wares made from the common clays.

[1] The Clay Resources of Southern Saskatchewan, by N. B. Davis, Mines Branch, Ottawa.

New Brunswick

Newcastle.—The clay bed lying beneath the coal seam in the vicinity of Newcastle is very soft and plastic and may be used as a pottery clay. The clay is not uniform in quality but varies slightly in different localities, as shown by the results of tests given in another section of this report. The clay contains a few coarse rock particles, and an occasional thin parting of coal, so that it would require to be washed and screened before using. The washed clay has good tensile strength in the raw state and is very smooth, so that it works well on the potter's wheel. The pottery forms made from it become hard at a comparatively low temperature.

Matt glazes maturing up to cone 3 can be used safely, but good pottery covered with bright glazes was obtained when burning to cone 05. This clay can be more easily treated by the washing process if it is dug and allowed to weather for several months.

St. John.—The red surface clay found in the vicinity of St. John and used in some places for making building brick gives very good results as a pottery clay. It is used at the Foley pottery on Loch Lomond road for making flower pots. When tested in the laboratory it was found to be one of the few surface clays which could be used on the potter's wheel without preliminary washing and screening, and a great range of glazes can be applied to the burned body when fired at temperatures ranging from cone 010 to cone 05.

Black ware can be made from this clay by packing the pieces in sawdust placed in saggers. After being fired to cone 06 the ware comes out with an ebon black colour throughout.

Use of Underglaze Colours on Pottery and Tiles

Some of the highly calcareous clays found in various parts of southern Ontario are very good for pottery, as far as working qualities are concerned, but some of them, owing to their high lime content, and the very porous character of the burned body, are difficult to fit with a suitable glaze, or to keep the glaze from sinking into the body during the firing.

It has been found that by dipping the green ware in a slip made with underglaze colour mixed with stoneware clay and then fired, that the coloured slip forms a hard surface. The burned ware is then covered with a transparent coloured glaze and burned the second time. Some very beautiful colour effects were obtainable in this way as the underglaze colour is not entirely concealed by the overglaze.

Underglaze colours are extensively used on pottery and tile made by the classes in manual arts at the Ottawa Public Schools. The underglaze colours are mixed with stoneware clay slip and applied with a brush to the incised or sunk patterns on the tiles and vase forms before they are quite dry. This is a simple method of obtaining colour effects on burned clay which answers very well for educational purposes, but if the piece is to be used it must be subsequently covered with a transparent glaze, as the slip colours, although hard, soon become soiled by handling.

New Potteries

We have frequently called attention to the opportunity that exists in Canada for selling locally made pottery and tile instead of imported

articles. People visiting places of interest desire especially to obtain articles of local manufacture which are distinctive in treatment and different from the monotonous and uninteresting mass production of foreign factories.

Three potteries were recently started with the object of taking advantage of this demand.

In Alberta a pottery at Banff is making wares in which pre-historic Canadian Indian designs are adapted for the decoration or the form. The stoneware clays from Saskatchewan are used, as there is no clay available at Banff for this purpose. As the mountain scenery and hot springs at Banff attract a multitude of visitors, it is one of the best points in Canada to display local handicrafts for sale.

A plant at Bowmanville, Ontario, has begun making garden pottery, such as jardinieres, window boxes, etc., using local clay, which burns to a hard buff coloured body and is either finished plain or covered with coloured glazes.

The Charlotte County Cottage Crafts, at St. Andrews, N.B., recently added pottery to their list of handicrafts. This department is under the superintendence of Mr. B. Kane, an English potter. Their object is to make clay wares of simple design and patterns, which cannot be obtained in the ordinary industrial factories. A red burning brick clay from the bank of the St. Croix river is being used at present, but other clays from New Brunswick, or Nova Scotia, will also be used.

IV

KILN SCUM ON FACE BRICKS

Occasionally red brick is much disfigured by light patches on the surface. This bleaching or discoloration, referred to as dryer, or kiln scum, is usually caused by the presence of soluble sulphates in the brick mixture, introduced by the clay or shale, or in the water with which the batch is mixed. During the drying process these soluble salts are concentrated at the surface of the bricks, and later, in the heat of the kiln, react with the iron oxide contained in the brick body causing it to impart a buff colour to the brick instead of the usual red, or the salts may simply form a light coloured opaque film which masks the true colour of the brick.

A bad case of scumming of face brick at their plant caused much trouble and loss to Merkleys, Ltd., at Ottawa, during 1921. Late in the season samples of scummed brick were submitted to the Mines Branch with the request that the source of the trouble be located and, if possible, a means of preventing further scumming be prescribed.

This brick is made from a mixture of Utica-Lorraine shale and marine clay by the stiff mud process. A qualitative analysis of the unburned brick showed the presence of water-soluble sulphates of lime and magnesia in sufficient quantity to account for the scum on the burned ware.

Several samples of shale and clay were then tested for sulphates with the following results. The clays were found to be free from sulphates, but, in all cases, the shale contained such salts. The upper beds of shale,

which are much weathered and have been leached by surface waters, contained very little, as did the lowest unweathered shale, while the middle beds which were only partially weathered and from which the sulphate salts produced by the oxidation of pyrite had not been removed by surface water, contained a fairly large percentage.

Since these middle beds are necessarily the source of most of the shale used, it would therefore not be practicable to work on a brick mixture without introducing these trouble-causing sulphates.

The problem then was to determine the best and cheapest means of rendering the sulphate salts harmless.

The water contained in the freshly made brick takes the soluble sulphates into solution, and as the drying process progresses the water flows by capillary action to the surface where it is evaporated, leaving its burden of soluble salts concentrated at the point where they are most harmful.

By the addition of barium carbonate to the brick mixture a reaction takes place between it and the soluble sulphates of lime and magnesia, producing barium sulphate and carbonates of lime and magnesia, all of which are relatively insoluble and therefore not productive of scum.

On analysis, the brick mixture yielded ·023 per cent of sulphur trioxide. By calculation it was found that about three pounds of barium carbonate per 1,000 bricks would be required to completely render the soluble sulphates insoluble.

Test bricks were made up in which this proportion of barium carbonate was incorporated, and burned in the Ceramic Laboratory. They were free from scum. Following this, a trial run of 30,000 bricks containing the barium carbonate was made at the brick plant. These bricks, when burned, were likewise free from scum.

Subsequently, the company installed a machine at the plant to feed the barium carbonate at a uniform rate into the pug mill, and it has been reported that during the early months of 1922 there has been no further trouble from scum.

Other salts of barium might be used in place of tbe carbonate, but they would have to be under much closer chemical control, lest they themselves cause scum.

Another means by which the scum may be prevented is to greatly increase the rate of drying the green brick. If the drying progresses slowly the evaporation takes place at the surface of the brick and any soluble salts will be concentrated there. If, however, the drying is hastened beyond a certain rate, the surface of the brick quickly becomes dry, and the balance of the moisture in the body is evaporated from below the surface and thus the bulk of the soluble salts will remain below the surface where they cannot produce any scumming.

This method has certain disadvantages. Some clays cannot withstand rapid drying without cracking. At best, surface checking is almost sure to develop. Then, also, in order to obtain rapid drying, alterations in the drying chambers would be necessary in some plants and additional fuel would be required where direct heat is used for drying.

V

WORKING STONY CLAYS FOR BRICK AND TILE

Clays containing pebbles or stones are usually avoided by brick and tile makers when choosing a suitable location for the erection of a plant.

Mistakes have frequently been made in the past in Canada by erecting plants on sites where stony clay occurred, when a thorough preliminary examination of the ground would have proved the futility of such an enterprise.

In some cases however, plants which have been working for years on perfectly stoneless clay, making good burned products, find on extending their clay pits that they are gradually running into a stony variety of clay.

In many parts of southwestern Ontario only a foot or two of the surface clay is stoneless and fit to be used for tile making, as stones invariably are encountered below this depth. A thin sheet of stoneless clay is so quickly worked out that the distance from the clay pit to the machine is constantly increasing, so that tilemakers would like to dig deeper in the vicinity of their plants if they dared to do so.

Over the recently opened agricultural region along the National Transcontinental railway, between Cochrane and Hearst, is a great sheet of plastic stony glacial clay. It is difficult to select a patch here which would be sufficiently free from pebbles for use in brick and tile making, but doubtless enough clean clay could be obtained for a small output. The clay burns to a buff colour, and will make brick or tile of very similar quality to buff or so-called white products in southern Ontario.

There are large masses of plastic yellowish-grey, glacial clay in the vicinity of Hamilton, Ontario, and along the Welland canal, as well as several parts of the Niagara peninsula. Much of this clay appears to be stoneless on a casual inspection, but close scrutiny generally reveals the presence of small rock particles and pebbles. The pebbles being coated with clay are liable to be overlooked in a hasty examination.

If a clay contains only a few scattered pebbles of rock other than limestone or dolomite it may be worked, if its properties otherwise are good. Broken wires may frequently occur in the cutter if stiff mud brick is being made, or an occasional brick will fire check if it contains too large a pebble, but if proper crushing rolls are provided for the clay to pass through, or if a dry pan is used for grinding, the loss of time, due to broken wires, can be stopped.

It is the presence of limestone pebbles that causes the real difficulty in working stony clays.

Palissy, the celebrated French potter, experimenting with various clays in the sixteenth century, and recording his impressions of them, says:—

> There are some kinds of clay which are of evil nature, because among them there are little stones, which when the vessels are baked, the little stones which are in the said vessels are reduced to lime, and suddenly when they come to feel the humidity of the air they swell and cause the said vessel to split in the place where they are enclosed, and this is because the said stones were calcined in the baking, and by this means many vessels are lost, however great the labour one may have employed upon them.

This quotation expresses the experience of all clay-workers from Palissy's time to the present day.

Crushing and Grinding

Many methods have been tried for eliminating the troubles due to limestone in clay, the simplest probably being that of using a pair of rolls set above the machine.

Rolls designed for crushing stony, plastic clays, reject the rocks or hard portions which are too large to fall within the angle of nip, and grind or pulverize the smaller stones which pass through with the clay.

Another device is to use a dry pan with the mullers set an inch or so above the surface of the pan. This process is supposed to break down the clay and force it through the perforated bottom of the pan while the pebbles stay behind. The pan is stopped occasionally and the stones thrown out.

Another plan is to grind the clay, pebbles and all, in the ordinary way in a dry pan, working continuously, and run the clay over screens so as to reduce the limestone pebbles to a small enough size to be harmless.

In using the dry pan method the clay must be in the dry condition, which involves storage room for drying clays, or else a rotary dryer should be used.

It has been proved that unless limestone in clay is ground finely enough to pass through a 30-mesh screen it will cause trouble by developing soft white specks in the burned bricks, which will cause flaking of the surface. Of course, these grains of lime, unless very plentiful, would not weaken a well-burned brick very materially, yet they would be sure to disfigure brick for facing purposes.

As it is not practical in any of the crushing or grinding processes outlined above to reduce the pebbles to this small size, the trouble with lime grains is not entirely eliminated by them. Furthermore, any of the rough grinding processes, such as passing the clay through rolls, merely accentuates the trouble, since it often breaks up one pebble of limestone into several smaller ones, each of which then becomes active after burning.

Washing

One effective way to treat clays containing limestone pebbles is by a washing process, which would leave the stone behind as a heavy residue and the overflow taking the fine clay down to settling basins.

Washing of high grade clays, such as china clay and paper clays is the common practice, but the poorer grade of clays used for brick and tile are not washed because the value of the product will not offset the extra expense. One plant at least in the United States successfully cleanses glacial clay from limestone pebbles by a washing process; it is situated at Hutcheson, McLeod county, in Minnesota. Its operation is described in the report by Mr. Frank Grout, on the Clays of Minnesota, Bulletin 678, United States Geological Survey, as follows:—

> The washing machinery occupies a space not over 20 feet square and 15 feet high, and washes 130 yards of clay in a day. The clay from the bank is hauled by cable car to the washer, where it is mixed with an excess of water and agitated by a series of vertical rods fastened to a rotating cross-beam. The harrow-like motion of these rods tends to throw the larger pebbles towards the centre and leaves the fine clay and sand suspended and distributed throughout the washer pit. A bucket elevator of continuous operation dips into the pit near the center and removes the gravel. The gravel, if cleansed, forms a by-product of considerable value. At the sides of the pit a screen of proper mesh allows the escape of the fine sand and clay to one of a series of open ponds in which they are allowed

to settle. After a time some of the water is pumped off and the rest is left to sink into the ground. The sand naturally settles close to the intake of the pond, and the clay is carried to the farther side. After partial drying the material is taken to the stiff-mud machine, where the clay and sand are mixed in approximately the same proportions in which they existed in the drift before the washing. Experiments are now in progress to determine whether the clay is improved by standing in the settling ponds all winter. The gravel is sold for concrete. Both the clay and the sand contain a considerable amount of calcium carbonate, but if care is taken to remove the coarser sand the lime does no harm, and it is certainly less abundant than in the unwashed drift. The plant at Hutcheson uses three round down draft kilns, and plans are made to double the capacity. It has been found possible with this clay to produce a very good drain tile and hollow building block, so that the production of common brick has become secondary.

Some of the glacial stony clays in southern Ontario would yield good tile clays when washed, and a small or medium sized local plant situated in a district where there are no tile plants at present and no stoneless clays available, should have a chance of succeeding.

A brick and tile plant at Perth, in eastern Ontario, made a practice of washing stony clay, which enabled it to produce durable wares which brought a higher price than the ordinary brick yard obtained.[1] The washing was done in the fall when brickmaking has ceased and a deposit of washed clay is ready for the following season's work.

In some cases the stones are loosely held in the clay and separation by washing is comparatively easy, but some boulder clays are so tough, and the stones so firmly embedded, that it is almost impossible to break up the clay and separate the stones by washing.

Such a plant, however, could not ship its products over any considerable territory as it would then meet the competition of plants using clay that is free from limestone pebbles and therefore more cheaply worked.

Burning

Hitherto we have been dealing with the treatment of the raw clay, but there are certain aspects of the burning and of methods of dealing with the burned product which must be considered.

In certain parts of England where clays containing limestone pebbles are worked it is customary to grind the clay as fine as practical, and after the bricks are burned the cars on which they are loaded are drawn through a pool of water so that they are completely immersed and saturated.

This treatment slakes the lime particles quickly and appears to do less damage than if they are allowed to slake and expand slowly from the moisture absorbed in the atmosphere.

At the Don Valley Brick Works, in Toronto, there are certain beds of stony clay interbedded with the stoneless clay, and both are worked together. As the clay drops through the rolls from the pug mill the larger stones are expelled by the rolls, and the small ones go through to the machine and are crushed. This clay is used in the manufacture of end wire-cut brick and burned in an overhead fired continuous kiln. The limestone particles give surprisingly little trouble in the burned product. It seems as if the method of burning had something to do with checking the subsequent activity of the lime particles due to the smoky atmosphere and reducing conditions present in the chambers of the continuous kiln. The reducing condition at high temperatures appears to

[1] Bureau of Mines, Ontario, Vol. XV, Part 2. Clay and the Clay Industry of Ontario., p. 76. M. B. Baker.

cause fusion between the surface of the lime particles and the surrounding clay, and this fused skin prevents the access of moisture to the lime; but whatever the reason is the lime is more effectually killed than it would be after coming through a well oxidized firing.

A plant producing face brick from glacial clays containing a few scattered pebbles of limestone is situated at Rymal, about 4 miles southwest of Hamilton, Ontario. The clay is ground in dry pans, screened, and made up into stiff mud brick. The burning is done in round down draft kilns up to as high a temperature as the bricks will stand without adhering. The product is a hard sulphur-coloured brick which shows no bad effects from the lime grains. In this case the lime grains probably form a fused bond with the clay so that there are no after effects.

Finally we have to consider the addition of some substance to the clay which would prevent the slacking of the lime after burning.

The writer has made many experiments to this end, but found that the only practical method was to add some salt to the water used in tempering.

A set of test pieces of burned clay containing limestone particles to which one per cent of salt was added, have now been standing in the laboratory for about a year and show little or no indication of disintegrating, while similar test pieces made up without the salt have gone to pieces long ago.

Pulverizing

The most satisfactory way to deal with stony clay containing limestone is by pulverizing it finely so that it will pass a 30-mesh screen.

This process involves drying the clay as a necessary preliminary step before pulverizing and this is done by passing the clay through a rotary dryer, or by storing it until it becomes dried out.

Some of the brick and tile plants in Indiana and Illinois, where there is a large demand for these products and where nothing but stony clays are available, use the pulverizing process successfully.

In the *Clay-Worker* for February, 1922, Mr. George M. Krick, of Decatur, Indiana, gives a description of a large plant in which he works stony clay. The clay is excavated with a ditching machine or clay digger, so as to take up the clay in thin layers and not in large lumps, so that it will dry faster. It is then passed through a rotary dryer 7 feet in diameter and 70 feet in length, that has a capacity of 30 tons per hour. The clay from the rotary drier is fed into a ring hammer pulverizer, and then elevated to screens.

Two gravity screens are used. The top screen has eight meshes to an inch and removes all the coarsely ground clay and gives greater capacity to the finer bottom screen, which only allows the dust and particles finer than 30 mesh to pass through. The pulverized clay is run into storage bins and is elevated from these to bins set above the pug mills, and the ordinary processes of brickmaking follow.

Another operator in the same region stores the clay for about six weeks before using. It is then run through a disintegrator and over an 8-mesh screen. The tailings from the screen are put through a Williams pulverizer.

VI

TENTATIVE METHOD FOR SAMPLING CLAY DEPOSITS

The following method of sampling shale and clay deposits is approved by the Committee on Standards, American Ceramic Society:—

(1) *Preliminary Sampling.*

A face of the body of clay to be sampled shall be carefully stripped of loose or foreign materials, and a series of parallel trenches cut, each a straight line, so as to make as nearly as may be a vertical section entirely across the outcrop. In the case of stratified or bedded deposits which are inclined, the direction of these trenches shall be, preferably, across the dip, so that a proportionate amount of clay will be obtained from each individual layer. Each trench shall be about 12 inches wide, and deep enough to produce at least 100 pounds of material. Where natural outcrops are not available for sampling, preliminary trials may be made with a hand auger, and test pits dug to the necessary depth to expose a section (or face) of the clay.

As this may yield more material than is necessary for even a complete series of tests, the clay from the different trenches shall be reduced to lumps not exceeding 2 inches in diameter, mixed together and reduced by quartering to about 100 pounds, which is to be sent to the testing laboratory. The quartering shall be done on a heavy sail cloth at least 8 feet square.

If the deposit shows distinct difference as regards structure, colour or texture, each bed showing these individual differences shall be sampled separately, provided these beds are sufficiently thick to be mined by themselves or can be thrown out if undesirable.

Where the clay is stored in stock piles the sample may be taken from these **provided they are representative.** At least one-half of the sample shall be taken from the lower third of the pile.

In the case of those clays which are being purified a sample of both the crude material and the clay as prepared for the market shall be taken.

The samples collected as outlined above shall be placed in clean, tight-weave strong sacks and carefully labelled by means of two tags each bearing the proper identification marks. One folded tag shall be placed within the sack and the other securely attached to the outside.

(2) *Extended Sampling.*

After a clay proves satisfactory in the preliminary testing, the surrounding tract must be surveyed and systematically sampled. A topographical survey of the tract shall be made. Holes shall be drilled or dug through the deposit not more than 100 feet apart. A record shall be kept of the thickness of each stratum encountered. By plotting these results to scale, the shape, size and dip of the strata can be estimated.

A few larger holes shall be dug at the extremities of the property or at any points of doubtful value in order to get samples large enough for a complete series of tests. These shall be taken under the direction of "Preliminary Sampling".

VII

CLAY-WORKING INDUSTRY

The clay-working industry in Canada is engaged principally in the production of structural materials, such as brick and hollow building blocks, and drain pipes for agricultural purposes. These are made from the surface clays of glacial origin which are so widespread over all the country; but at a few points shales from the older rock formation are the raw materials used. Vitrified structural wares, such as sewer pipes, are made to some extent, but no paving brick are produced, mainly for lack of the proper kind of vitrifying shales employed in their manufacture. Four plants are engaged in the manufacture of heavy household pottery and

stoneware goods such as crocks and jars. Two of these plants import their clays from the United States, but the others use the domestic clays in Nova Scotia and Saskatchewan.

White vitrified floor tile are made at one point, and sanitary ware and electric porcelain at three others. All the industries making white bodies import their raw materials from both England and the United States.

The annual value of clay products made in Canada has been steadily increasing during the past few years, and is now almost up to the level of 1912, which is the highest record so far. The production in 1912 was much greater in proportion, but the cost of operation was much less at that time.

The following table shows the value of the principal classes of clay products manufactured in Canada for the year 1920, also the increase over 1919:—

	1919	1920
Bricks	$5,154,381	$6,840,533
Fireproofing and hollow blocks	462,582	940,422
Sewer pipe	1,074,146	1,549,090
Field drain tile	616,510	562,652
Pottery	184,474	209,171
	$7,492,093	$10,101,868

Special Clay Wares

Some of the existing plants throughout Canada, engaged in the manufacture of brick and drain tile, might take up with advantage the making of a more specialized and profitable line of wares wherever they have a suitable clay or shale. There is a large and growing demand for the ordinary red floor tile used in corridors, kitchens, hospitals, and other places, which may easily be made from some of the red brick shales by simply grinding them finely. While these shales may not be suitable for tiles for outdoor use, on account of their porosity, they will give good wear in indoor positions. Roofing tile, and tile for lining digesters in the pulp mills, are other profitable lines for the manufacture, but these would have to be salt glazed.

Porous clay wares, such as iceless refrigerators and water coolers, would have a large sale if any one undertook their manufacture. Vessels of this kind owe their cooling properties to their power to soak up a large quantity of water, and the evaporation of this water as it comes to the surface keeps the contents cool. There is a vast amount of buff burning clay near Lake Erie, and elsewhere in southwestern Ontario, used for making porous drain tile, which would probably be suitable for this kind of ware.

White Table Ware

No table ware, such as white earthenware, semi-porcelain, or china, is produced in this country, the reason probably being due to a dearth of the proper kind of raw material, and the market not being large enough.

The commoner kinds of table ware have hitherto been made so cheaply and so well in European countries that it was impossible for Canada to compete with them. In recent years, however, the prices for these goods have increased fourfold or even more, so that there is now a much better opportunity for manufacturing in Canada at a reasonable profit.

The manufacture of staple lines would be the safest business to undertake at the start. These include: (1) plain white, or, as it is sometimes called, white granite ware; (2) gilt wares, light and heavy, these being simply decorated over the glaze with a gold line, or with a stamped device in gold, the commonest on the market being the clover leaf, but the Canadian manufacturer should stamp his ware with a gold maple leaf or a beaver; (3) ware decorated with a transfer pattern in one colour, generally called printed ware; (4) water jugs, either plain white, or decorated with gilt band, or painted decoration.

There is an enormous quantity of these goods sold in Canada, and any manufacturer producing them at say 20 per cent less than the imported wares would be assured of the greater part, if not all, of the business.

Raw Materials.—A mixture of four kinds of raw material is used in making the above wares, these being: china clay, ball clay, flint, and feldspar. In addition to these, a supply of fireclay is necessary for making the saggers or containers in which the ware is burned.

The present supply of china clay in Canada is limited to one deposit, that of the Canadian China Clay Company, of Huberdeau, Quebec. This deposit is being developed by new mining methods, and a larger output of clay is promised than was formerly available. So far no ball clay has been discovered within reach of transport in Canada. Certain of the white sandstones in eastern Canada, when pulverized, can be used for the flint in pottery.

There is a plentiful supply of high grade feldspar for pottery bodies and glazes in eastern Canada. Any firm doing a large amount of business in white earthenware could purchase the crude feldspar and quartz from the mines, and do their own grinding.

Fuel.—As it requires six tons of coal to produce one ton of white table ware, the fuel question would have to be studied closely, and particular attention paid to design of kilns and method of fuel saving. Putting the coal through a gas producer and using the gas for fuel in a tunnel kiln seems to be the practice that is approved by most ceramic engineers.

Location.—The location for a plant making white table ware depends on various factors. The assembling of raw materials, the cheapest place to deliver coal, the transportation facilities for assembling and distribution, and the labour supply are the principal things to be taken into consideration.

Points on the St. Lawrence river up to Montreal have the advantage of ocean freight delivery of supplies of English china clay and ball clay without re-handling, as well as access to the Nova Scotia coal fields. Points on Lake Erie, such as Port Stanley or Port Colborne, are well situated for receiving supplies of coal and fireclay from the United States, and points on the Welland canal have similar advantages for assembling raw materials and distributing finished products.

VIII

BALL CLAY IN SASKATCHEWAN

We have frequently called attention to the occurrence of ball clays in the southern part of the province of Saskatchewan, and tests and analyses were published in 1918.[1]

[1]The Clay Resources of Southern Saskatchewan, Mines Branch, 1918, p. 15.

Professor W. G. Worcester, of the department of Ceramic Engineering, at the University of Saskatchewan, recently made tests on some ball clay beds, and the following is his report on a clay deposit near Willows:—

On Sec. 33, Tp. 7, R. 28, west of 2nd Mer., there is a section of clay approximately 30 feet in depth. The lower 10 feet is greyish white, somewhat stained with yellow along the cleavage lines. Close inspection reveals the presence of many minute iron concretions distributed throughout the mass.

The sample for this report was taken at an opening or cut from which two car loads of material had been taken for shipment to a brick plant a couple of years previous. Naturally a perfect sample could not be obtained due to cave-ins. The best average possible was taken, shipped to the ceramic laboratory at the University of Saskatchewan, where, up to the present, it has been tested as follows:—

Washing Test

	p.c.
Caught on 40 mesh	1·10
" 80 "	1·15
" 100 "	0·03
" 200 "	1·95
Total	4·23

The greater part of the residue caught on the screens is the iron in concretionary form, a little mica and some quartz grains.

The washed sample required 30 per cent of water to produce plasticity.

The drying shrinkage was 7·4 per cent, the trials dried safely without warping or checking.

Burning Shrinkage

	p.c.		p.c.
Cone 1 (2102 F)	5·9	total	13·3
" 3 (2174 F)	6·8	"	14·2
" 5 (2246 F)	6·8	"	14·2
" 7 (2318 F)	7·6	"	15·0

As a ball clay the above per cent of shrinkages are normal and well within commercial limits.

As a comparison two greatly used ball clays are listed.

English ball clay No. 12	total	16·2
Tennessee " 7	"	15·7

These clays were burned in the same kiln and at the same time as the Saskatchewan ball clay. Thus it can be seen that the latter falls in the same class, as far as shrinkage is concerned.

Burned Colour

In order that the colour might be properly guaged or classed, samples of English and American commercial ball clays were obtained. Trial pieces were made of each and burned in the same kiln with the Saskatchewan clay. When cool and removed from the kiln the several samples were arranged according to their respective degrees of whiteness or tints, resulting as follows:—

Saskatchewan ball clay washed	1	best
Tennessee No. 7 ball clay	2	next
" " 9 " "	2	"
Mayfield, Kentucky, ball clay	2	"
Tennessee No. 11 ball clay	3	"
English " 12 " "	4	"
Saskatchewan ball clay unwashed	5	"

In this test the Saskatchewan clay gave a better colour (light cream) than any of the imported standard clays. We can therefore say that it would prove highly suitable in the body mixtures for the manufacture of whiteware and porcelain.

Chemical Analysis

That the chemical properties of the Saskatchewan clay may be compared with the better grades of commercial clays, the following table is given.

——	Sask.	Mayfield, Ky.	S. Amboy, N.J.	Wareham, Eng.	Hall, Eng.
Silica	59·03	56·40	44·89	55·00	39·60
Alumina	25·88	30·00	37·27	29·71	45·00
Iron	0·94	.....	0·97	2·14	.....
Lime	0·24	0·40	0·41	0·62	0·10
Magnesia	3·23	5·27	0·19	3·44	3·30
Soda and potash	1·82		1·44		

It will be observed that the Saskatchewan clay corresponds quite closely with some of the best commercial ball clays of the world, and will without a doubt, prove equally as good in actual use.

The deposit represented by the sample under test is directly alongside the Canadian Pacific railway, and is four miles east of Willows station.

With the single exception, that of having to wash to remove the concretionary iron, the Saskatchewan ball clay is a valuable deposit. Its working properties are good and its burned or final colour ranks but very little below some of the china clays of the world.

Unquestionably, Saskatchewan has a valuable resource in this deposit of material. However, it should be borne in mind that there are undoubtedly other deposits of similar properties and value at other points in the province, but studies of these deposits have not yet been made.

The above clay, however, ranks among the very best of its kind in the province, being up to commercial requirements, and situated close to rail facilities.

ROAD MATERIALS DIVISION

Henri Gauthier

The investigation of road materials during the year covered the examination and sampling of a number of gravel deposits and rock quarries along the Ottawa-Point Fortune highway in Ontario, and the continuation of a road material survey in the Canadian National Rocky Mountains Park in connection with the construction of automobile roads by the Parks Branch of the Department of the Interior.

The object of these investigations was to supply the Ontario Department of Public Highways and the Parks Branch engineers, respectively, with information on the comparative qualities of the available supplies of roadmaking material in the neighbourhood of those highways, the surfacing of which is in contemplation.

In carrying on these two investigations about 25 samples of rock and gravel in Ontario, and over 80 samples of rock, gravel, boulder clay and natural subsoil in the Rocky Mountains Park were collected. These samples were examined and tested in the laboratories of the Division during the winter.

Detailed reports on these two surveys were prepared and sent to the Department of Public Highways of Ontario and to the Commissioner of Dominion Parks. In these reports are given the location and description of the more promising deposits along the roads in question as possible sources of road surfacing material. They also contain the results of laboratory tests on samples collected and a discussion on the comparative value of the various materials examined and tested. These results appear in tables in the following pages.

During the year a number of samples of stone and gravel which were submitted for testing were reported upon. Tests on samples collected in Nova Scotia in 1920[1] were completed. The results of these tests are recorded in the accompanying tables.

An abrasion test on concrete cylinders was also made, in compliance with a request from Mr. E. Viens, Chief of the Laboratory for Testing Materials, Department of Public Works, Ottawa.

This test was made to secure information regarding the comparative value of concrete mixtures made of different types of stone aggregate as to their resistance to wear. For such test, a machine on the plan of the brick rattler is used in the United States. It is called the Talbot Jones rattler.[2] As neither the Public Works nor the Mines Branch laboratories possesses such a machine, it was thought worth while to carry on an experimental test using the road stone Deval abrasion machine. While supplying Mr. Viens with reliable information, the test came under our study of methods of testing materials.

The writer was ably assisted by his confrere, Mr. R. H. Picher, both in the field and in the carrying out of laboratory work.

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[1] Mines Branch, Summary Report of Investigations in 1920, p. 72.

[2] A Method of Making Wear Tests of Concrete, by D. A. Abrams, Prod. American Society for Testing Materials, Part II, 1916.

Acknowledgment is also due to Messrs. R. S. Stronach, superintendent of Rocky Mountains Park, and J. M. Wardle, chief highway engineer, and their assistants, who spared no effort in facilitating the field work, in the matter of transportation, and by rendering other services with unfailing courtesy and kindness.

I

LABORATORY TESTS ON ROAD BUILDING STONE

Tests are made for two purposes—to determine in the materials tested, their characteristic qualities, and whether they conform to a certain previously determined standard established by the test of service. After it has been determined that a certain class of material under the conditions existing on any particular surface is the most suitable for the purpose for which it is employed specifications can be drawn up which will insure that material purchased in the future shall conform to the requirements. Then it can be determined by tests, what material is similar to the types which have proved satisfactory.

Laboratory tests have been devised to enable us to determine these qualities.

After making the tests that show the relative qualities of different kinds of rocks, the highway engineer is in a position to make future classification of similar materials largely by inspection.

There are five standard tests regularly made in the laboratory upon road building rock, which furnish a ready means of judging the comparative value of a rock as road metal. The most important are those for resistance to abrasion (percentage of wear) and for resistance to impact (toughness). The others are for hardness, specific gravity, and absorption.

As a result of comparisons made by engineers between laboratory tests and the wear of the stone in practice, certain limits have been set upon the values for the toughness and percentage of wear of stone that is to be used in macadam construction.

The American Society of Civil Engineers recommended in 1917 that stone used in waterbound macadam roads should have a percentage of wear of not more than 5, and a toughness value of not less than 6. The specifications adopted by the American Society of Municipal Improvement in 1914 require that stone used in the wearing surface of bituminous macadam or bituminous concrete roads shall have a per cent of wear of not more than 3·7 and a toughness of not less than 13. The United States Office of Public Roads sets the minimum limit of toughness for stone used on roads subject to traffic of less than 100 vehicles a day at from 5 to 9, except in the case of bituminous concrete, where the lower limit is 7. On roads subject to traffic of 100 to 250 vehicles a day, the minimum toughness is 10 for waterbound macadam and bituminous macadam, and 13 for bituminous concrete.

Tests Made upon Gravel

Because no two gravels have the same composition and character, engineers have found it difficult to draw specifications within narrow limits for road gravel. However, there are certain requisites which have been recognized as necessary to give satisfactory results.

Gravels are examined to determine their adaptability for the construction of gravel macadam or for concrete roads, and to obtain some knowledge of their ability to resist wear and to bind in a macadam surface; also to obtain information on the probable strength of concrete in which they might be used as the aggregate.

In the examination of gravels, estimates are made of the average composition of the pebbles, the relative proportion of the various sizes, the shape of the pebbles, and the nature of the impurities present. The tests made are, granulometric analysis, colorimetric tests for impurities, and mortar test.

Explanation of Tests

Granulometric Analysis.—The ability of gravel to pack well in a road bed is dependent to a large extent on the proportion of the sand content, on the amount of clay and silt present in the sand, and on the grading according to size of the particles making up the gravel and sand.

The results obtained from the granulometric analysis give an idea of the texture of the sand and gravel with regard to its grading in size of grain.

Sand, gravel and boulder clay deposits are variable in their grading from point to point. Consequently one analysis cannot determine the grading of the material. If the sample, however, has been carefully collected as representative of the average run of the bank the figures recorded in this test will furnish a general indication of the material available in the deposit in question.

General limiting values for the interpretation of granulometric analysis of road surfacing gravels has been adopted by the United States Office of Public Roads as follows:—

(*a*) All to pass a 1½-inch screen and to have at least 55 and not more than 75 per cent retained on a ¾-inch screen.

(*b*) At least 25 and not more than 75 per cent of the total coarse aggregate to be retained on a ¾-inch screen.

(*c*) At least 65 and not more than 95 per cent of the total fine aggregate to be retained on a 200-mesh screen.

From 8 to 15 per cent of the sand content in gravel should be clay or similar binding material such as oxide of iron. The binding or cementing value of gravel is also improved by an appreciable amount of calcium carbonate. Angular pebbles have a tendency to compact under traffic more rapidly than rounded pebbles and therefore are likely to produce a firm road surface in a shorter time.

Fineness Modulus.—The fineness modulus is a measure of the size and grading of the aggregate. It is the sum of the percentages given by the granulometric analysis, divided by 100.

This is determined by using the following sieves from the Tyler Standard series: 100, 48, 28, 14, 8 mesh. A well graded coarse sand up to 4-mesh sieve will give a fineness modulus of about 3·00. A fine sand such as drift sand may have a fineness modulus as low as 1·50. A coarse aggregate for concrete, graded from ¼ inch to 1½ inch, will give fineness modulus

of about 7·00; a mixture of the above with coarse sand of fineness modulus 3 in proper proportions for a 1: 4 mix will have a fineness modulus of about 5·80.[1]

Percentage of Clay and Silt.—In the method employed the percentage of clay and silt is determined by elutriation. A 100 gram sample of sand is placed in a glass vessel and subjected to a rising current of water of constant head, which carries off the silt and clay. When the discharge water becomes clear, the washed sand is dried and weighed. The loss in weight represents the percentage of silt and clay in the sample.

Gravel-Pebble Classification.—Experiments have shown that the percentages of wear increase with the amount of soft material in the gravel, and because of results obtained in comparing pebble composition and per cent of wear, the pebbles are divided into durable, intermediate, and soft types, according to their average toughness values and percentages of wear. The durable types include pebbles derived from igneous and metamorphic rocks having an average per cent of wear less than 4 and an average toughness over 12. The intermediate types include those with an average per cent of wear of from 4 to 5·6 and an average toughness of from 5 to 12. These are limestone, dolomite, slightly weathered rocks belonging to the durable types or igneous rocks with a high percentage of mica, hornblende, or other soft mineral. The soft types include rocks having a percentage of wear above 5·6, and thoroughly weathered stones of all kinds.

Mortar Tests.—Tensile and compression tests are made, according to standard methods, on a mixture of three parts of sand to one of cement, gauged with distilled water and using 2″ × 4″ cylinders for the compression test. Test pieces of mortar made of standard Ottawa sand are made with each set of samples and all are broken at the end of seven and twenty-eight days. The results for the sands under test are expressed in percentages of strength relative to that of the standard sand.

Although in many cases the strength of sand mortars, as expressed in terms of percentage of strength of the standard mortar, are lower for the twenty-eight day break than for the seven day, the actual strength in pounds per square inch are almost invariably higher. This is due to the fact that the increase in strength on ageing is greater in the standard mortar than in the case of these particular sand mortars.

An examination of the results obtained shows that the finer sands make poor concrete material. As a rule a sand which has 75 per cent retained on the 48 mesh, or a fineness modulus of about 3, will make good concrete.

Sands containing a percentage of clay and silt above 10 per cent proved that they would not make satisfactory cement mortars and would not be safe to employ in important work.

Specifications for fine aggregates to be used in cement concrete for pavements require that not more than 10 per cent of the sand grains below the ¼-inch size shall pass a sieve having 50 meshes to the linear inch, and not more than 2 per cent shall pass a sieve having 100 meshes to the linear inch.[2]

[1] Design of Concrete Mixtures, by Duff A. Abrams, Bulletin I, Structural Materials Research Laboratory, Lewis Institute, Chicago, 1919.

[2] Report of Com. III, 1914, Nat. Conf. Concrete Road Building, U.S.

Colorimetric Test for Organic Impurities.—The presence of certain organic matter in sands and gravels has a decided weakening effect on concrete made from them.

A colorimetric test for organic impurities has been devised at the Structural Materials Laboratory, Lewis Institute, Chicago, and is described as follows.[1]

A sample of sand is digested at ordinary temperature in a solution of sodium hydroxide. If the sand contains certain organic materials, thought to be largely of a humus nature, the filtered solution resulting from this treatment will be found to be of a colour ranging from light yellow through the reds to that which appears almost black. The depth of colour is measured by comparison with proper colour standards. The depth of colour produced by digesting a sand with a 3 per cent solution of sodium hydroxide has been found[2] to bear a relation to the compressive strength of mortar made with this sand and is about as follows:—

Colour values	Reduction in compressive strength of 1: 3 mortar per cent
250	10—20
500	15—30
1,000	20—40
2,000	25—50

II

REPORT ON THE INVESTIGATION OF A NUMBER OF ROCK QUARRIES AND GRAVEL DEPOSITS IN PRESCOTT AND RUSSEL COUNTIES, ONTARIO

In the early part of the summer of 1921, in compliance with a request from the Department of Public Highways of Ontario, an investigation of road materials was carried on in Russel and Prescott counties.

Because of the lack of available staff and want of time a complete survey could not be conducted by the Road Materials Division. However, one week was spent going over some of the more important deposits of stone and gravel, considered as possible sources of road-building material for the surfacing of the Ottawa-Point Fortune highway or the construction of main county roads.

In carrying on this work, information supplied by Mr. F. A. Senecal, county road superintendent, and Mr. F. Brinkman, resident engineer in charge of the Ottawa-Point Fortune highway, as to the location of the deposits to be examined, was made use of.

The work consisted in examining the deposits, in estimating the probable amount of available material, and in taking samples for examination in the laboratory of the Division.

ROAD MATERIALS AVAILABLE

Bed-rock and gravel constitute the materials in Russell and Prescott counties which can be used in road construction. The bed-rock consists of Trenton and Chazy limestones.

Outcrops of limestone and deposits of gravel are not plentiful, but they are found at intervals, and constitute a sufficient supply for local road work,

[1] Colorimetric test for Organic Impurities in Sands, by Duff A. Abrams. Circular No. 1. Structural Materials Research Laboratory, Lewis Institute, Chicago, 1917.

[2] 1916 Report of Committee C-9 of the American Society for Testing Materials.

Bed-rock

Because of their more general occurrence along the Ottawa-Point Fortune highway, and because of the ease with which they can be quarried, limestones have been and will probably be the chief source of road material for that main provincial road.

These limestones, however, are of only moderate durability and are not suitable for roads subjected to heavy traffic, except when used with a bituminous or portland cement binder.

The durability of limestones varies with their composition and texture. The finer and more even-grained varieties, as a rule, are the more durable. A tough, hard limestone can give as satisfactory results in some cases as the most durable igneous rock, but the great majority of limestones are soft and wear rapidly. Most of the limestones, however, cement with ease in a road bed, and under light traffic conditions have given good service.

The best road-making limestone is probably that of an even, fine-grained texture, which is unweathered, and which contains a minimum of black shale and of clayey partings so frequently encountered in that kind of stone.

The limestones occurring in Russell and Prescott counties are of medium hardness, and the result of tests on their physical properties compare with the average values obtained for stone of the same geological formations occurring in the neighbourhood of Ottawa.

Stone from the various deposits examined has been used to a certain extent in the construction of waterbound and bituminous macadam roads, on some stretches of the Ottawa-Point Fortune highway, and within the limits of a number of towns. It has made fairly good roads under light traffic conditions, but roads built with similar stone in the vicinity of Montreal and Ottawa, and carrying a heavy automobile traffic, have proved to wear away fast even where a bituminous binder is used.

A description of a few of the more important deposits of limestones visited is given below. In Table I are given the results of laboratory tests performed upon samples collected from these deposits.

Description of Bed-Rock Deposits

I. Paradis Quarry, Plantagenet

Location.—Lot 9, con. III, North Plantagenet township. Between main road and South Nation river, about 1 mile west of Plantagenet.

Description.—The quarry lies on the edge of a ledge which forms quite a scarp facing the South Nation river to the north. The size of the excavation is small, but much stone is available. The exposure above the actual quarry floor consists of 10 to 12 feet of limestone.

The stone in the upper 5 feet is thin-bedded, flaggy, and contains shaly partings. It is uneven in texture, varying from fine to medium-grained, with secondary calcite crystals. It is dark-coloured. Some of it is weathered, but the greater part is fresh.

The lower stone occurs in thicker layers and is of a more uniform texture.

These beds are nearly flat-lying; they dip but very slightly to the south. Some of the upper layers are much shattered, but the lower beds are only moderately jointed. The overburden is from 1 to 2 feet thick, but apparently becomes thicker away from the edge of the outcrops towards the south. A 20-foot working face could be exposed and large quantities of stone obtained without drainage trouble. See sample 1, Table I.

II. Lambert Franche's quarry, 2 miles south of Wendover

Location.—Lot 32, con. II, North Plantagenet township.

Description.—Stone has been quarried to a small extent for road-building material from an important ridge of limestone, which can be traced for several miles eastward. The ridge is for the most part thickly covered, but in places outcrops are seen on its northern slope with good opportunity for quarrying.

On the Franche farm, a small quarry has been worked in the upper part of the steep slope, just south of the concession road. The top of the ridge at that point is over 75 feet above the flat to the north. In the excavation not less than 20 feet of limestone is exposed, and a working face much higher than that could be had along the cliff.

The stone resembles somewhat that in the Paradis quarry, near Plantagenet, except that it contains less shaly partings. It is thin-bedded, flat-lying, dark-coloured, fossiliferous limestone. It is uneven in texture, with thin, wavy, irregular streaks of shaly material. The stone is fresh, except in the upper 2 to 4 feet, in which it is much shattered and weathered. The amount available here is practically unlimited. It is close to the road. See sample 2, Table I.

III

Outcrops are also seen farther east along the ridge above described, on lot 13, con. III. The limestone is specially well exposed on the farm of L. E. McCormick, in a small opening near the top of the ridge, just east of the road, and in a cliff at least 25 feet high along the escarpment a short distance from the road. The stone exposed near the road is fresh and resembles that seen in the Paradis quarry. The material exposed in the cliff is thin-bedded and uneven in texture. It is much shattered and not as fresh as the stone outcropping near the road and on top of the ridge.

Large quantities of limestone can be easily quarried here.

IV. J. Brisebois' quarry, Alfred Centre

Location.—Lot 6, con. IV, Alfred township.

About 2 miles north of the Ottawa-Point Fortune highway.

Description.—The quarry, which is only a few hundred feet south of the concession road, is located on the north side of an extensive ridge of limestone over 1½ mile long, running nearly east-west. There are many outcrops which could be developed along this ridge. The stone has been quarried to a small extent at several points, but the Brisebois quarry is the most important. It is opened in the face of the ledge, which is over 25 feet high at that point. The quarried wall was over 100 feet long and

not less than 15 feet high at the time of our visit. Quarrying operations had just been started, and large quantities of stone for road purposes were to be obtained during the summer.

The character of the limestone is dark blue, fine-grained, with small crystals of brown calcite. It is hard and brittle and breaks very irregularly. From bottom to top of the exposure the stone is uniform. It is fresh and occurs in massive beds from 6 inches to a couple of feet thick. Vertical joints are frequent. The material in the upper 2 or 3 feet is shattered and somewhat weathered. Large quantities of stone are available, with fair opportunity for quarrying. See sample 3, Table I.

V

At the west end of the deposit above mentioned, the stone has been quarried to a small extent on the farm of Jas. P. Lett.

Just west of the north-south road, massive beds of fresh stone are exposed to a depth of 10 feet in a small opening, and outcrops are seen over a width of 125 feet.

VI

On lot 8, on the farm of Jos. Robillard, there is an old quarry by the road. Quick lime was produced there at one time. The quarried wall is 10 feet high, but a working face of twice that height is possible. The stone resembles in character that in the Brisebois quarry; it belongs to the same ridge.

VII. House quarry, L'Orignal

Location.—Lot 20, L'Orignal, Longueuil township.

The quarry lies about 1½ mile southwest of the town of L'Orignal, on the west side of the Cassburn road and is opened in the face of a bluff overlooking Hill creek.

Description.—The bluff is about 50 feet high in places, and is continuous for quite a distance in a northeast direction. This is on the western edge of a plateau extending to the east. The upper layers have been worked for building stone and road material to a depth of 10 feet over an area of 35 yards by 20 yards.

Much variation is noticeable in the character of the stone exposed. It ranges from fine, even-grained, to uneven, coarse-textured and highly fossiliferous limestone. It is, as a rule, fresh and of a dark bluish colour. The stone occurs in irregular layers, varying in thickness from 1 inch to 1½ feet. The bedding is much disturbed, and joints are frequent at various angles. The amount available without drainage trouble is large.

Stone from here was used in the construction of a macadam road through the town of L'Orignal. That road was in fairly good condition last year after several years of service.

Laboratory tests have shown that this limestone is a comparatively durable road metal. See sample 4, Table 1.

VIII. James Ross quarry, Little Rideau

Location.—Lot 28, con. I, East Hawkesbury township. This is an old quarry located in a bush immediately west of the crossroad and about 1 mile south of the Ottawa-Point Fortune highway.

Description.—About 15 feet of limestone are exposed in an excavation 75 yards by 50 yards. The stone is thick-bedded, coarse to medium-grained, bluish-grey and highly fossiliferous. It contains numerous calcite crystals, breaks very easily, and is rather soft for road purposes. The beds show little weathering, are horizontal and not much jointed. Geologically they belong to the Chazy formation.

The deposit forms an extensive ridge running northeast across the road, and is partly covered with bush and a thick overburden. Large quantities of stone have been obtained here in the past, particularly for purposes of heavy construction. The amount to be had is still large. The surface beds have been worked over a large area, and many thousands of cubic yards of stone can be obtained from piles of debris. The probable value of this stone as road metal is closely comparable with the rock in place. See sample 5, Table 1.

TABLE I

Results of Laboratory Tests upon Limestones

Sample No.	Locality	Physical Properties						Remarks
		Per cent of wear	French coeff. of wear	Toughness	Specific gravity	Absorption in lbs. per cu. ft.	Hardness	
1.....	Paradis quarry, Plantagenet.....	4·24	9·4	8	2·71	0·24	14·4	Dark grey, fine to medium-grained limestone.
2.....	Franche quarry, Plantagenet.....	4·67	8·6	8	2·71	0·22	14·5	Dark grey, fine-grained limestone, with shaly partings.
3.....	Brisebois quarry, Alfred Centre..	4·37	9·2	4	2·70	0·18	15·0	Brownish grey, fine, uneven-grained limestone, brittle.
4.....	House quarry, L'Orignal.....	3·36	11·9	7	2·70	0·47	15·2	Light brownish grey, medium-grained uneven textured limestone, with calcite crystals.
5.....	Ross quarry, Little Rideau.....	4·20	9·5	5	2·72	0·64	11·0	Light coloured, coarse-grained limestone, with calcite crystals.

It is obvious from the results contained in Table I that while the majority of the limestones tested comply with the specifications for the physical properties of broken stone for waterbound macadam with light traffic, they fail to show sufficient ability to resist wear and impact required in bituminous macadam or bituminous concrete, or even in waterbound macadam with a traffic of over 100 vehicles a day.

As a class, limestone of the type occurring in Russell and Prescott counties is a rather poor road metal, but it should not be rejected as unsuitable for road purposes. Local conditions have to be taken into account, and because it is the only available class of bed-rock in that district, it can be economically used if the roads built of it are properly maintained. In areas where gravel is scarce, this stone can be advantageously used as coarse aggregate in concrete mixtures for the construction of concrete roads. However, the best results cannot be expected with the use of this stone in concrete pavements. Service and laboratory tests on concrete have shown that the nature of the stone aggregate used in a concrete mixture has some influence on its resistance to wear, especially if the concrete is subjected to continuous and severe abrasive action. In practice, a French coefficient of wear of 10 is considered as a minimum value for stone to be used in concrete roads.

GRAVEL

In the following pages the character of the deposits of sand and gravel visited is briefly described. Such information as can be gathered from laboratory examination of samples upon the suitability of the material for road surfacing or concrete purposes can be obtained by referring to Tables II and III.

On account of the short time available, only a few of the deposits of gravel were examined. There are many other deposits in that district from which suitable material may be had. Gravel is cheap in first cost, and easily handled. It is especially suitable for the improvement of county or secondary roads, as a gravel macadam is more economically constructed and easier to maintain than a broken stone construction.

Description of Gravel Deposits

PRESCOTT COUNTY

I. Belanger pit, Point Fortune

Location.—Lot 1, con. I, East Hawkesbury township. Pit located along southern edge of deposit, about one mile south of the town and immediately west of cross road.

Description of deposit.—The deposit forms an extensive ridge running in an east-west direction; it covers an area of approximately one mile by one-quarter mile and rises to about 25 feet in places.

The amount excavated is 25 yds. × 15 yds. × 4 yds. = 1,500 cu. yds. Maximum height of wall = 12 ft.

Character of material.—The exposure about the centre of the main wall which faces south, is as follows from top to bottom of bank:—

(*a*) Overburden and weathering part.....3 to 8 feet thick.
(*b*) Fine gravel and coarse sand.....2 feet.
(*c*) Coarse gravel.....3 to 5 feet.
(*d*) Fine gravel coarser than (*b*).....2 feet.

Balance covered with talus.

Samples of layers (*b*) and (*c*) were collected.

The weathered part contains yellowish loam. There are very few boulders over 6 inches in size, but coarse gravel occurs in larger proportion than fine gravel. The pebbles are round in shape and composed of granites, gneisses, quartzites, limestones and sandstones. The material is fairly clean, but in certain layers weathered pebbles are found in large quantity. The amount of gravel available in this deposit is considerable, but not without a good deal of stripping. The overburden is apparently thick and there are also a few trees on top of the deposit.

The actual pit has been worked in the past in a careless manner.

See the results of analyses of samples 1 and 2 in Tables II and III.

II

Exposures of gravel can be seen on the northern edge of the ridge above described in cuts along the road leading to the C.P.R. station just south of the town of Point Fortune.

The deposit, which extends also to the east of the road, forms quite a hill here. From 6 to 8 feet of gravel can be seen. One sample was taken. See No. 3 in Tables II and III.

III. Oscar Cousineau's pit. West of Point Fortune

Location.—Lot 9, con. I, East Hawkesbury township. The pit is situated in a pasture about one-half mile south of the main road.

Description of deposit.—Lense shaped deposit approximately 150 yards wide by 300 yards long, trending northeasterly. The deposit rises only very slightly above the general level of the surroundings. The gravel has been excavated to a depth of 9 feet and the actual size of the pit is 65 yards by 35 yards = 6825 cubic yards.

Character of material.—Medium gravel, carrying very fine sand. At the northwest corner of the excavation clean yellow medium sand occurs while gravel with all pebbles under 1½ inches in size is seen in the northern bank. Good fine gravel, interbedded with fine yellow sand is exposed along the east wall of the pit. The walls on the west side are covered with talus. The overburden, which consists of gravelly loam, is thin but some weathering is noticeable to a depth ranging from 2 to 2½ feet. Several thousand cubic yards of material are available from this deposit with very little stripping, but there is a long haul to the main road over a rough farm road.

One sample of the bank run on the east side of the excavation was taken. See No. 4 in Tables II and III.

IV. Dan. Kirby's pit, Chute à Blondeau

Location.—Lot 17, Broken Front concession, East Hawkesbury. Pit located just north of main road.

Description of deposit.—Small deposit forming a knoll near farm buildings. About 6 feet of gravel is exposed in an irregular shaped excavation at the southwest corner of the deposit. Size of pit: 35 yds. × 30 yds. × 2 yds. = 2,100 cu. yds.

Character of gravel.—Bouldery gravel. Many large boulders over 2 feet in diameter lying in pit. The gravel contains a large percentage of material between 2 inches and 6 inches in size. There is very little stripping to be done, but section of exposure shows much weathered gravel carrying loam.

The amount available is small.

One sample representing the finest material in the bank was taken. See the results of analysis of sample No. 5, Tables II and III.

V. Dandy pit, 5 miles west of Little Rideau

Location.—Lot 36, Broken Front concession, East Hawkesbury township. Pit located in a pasture, between the main road and the river shore.

Description of deposit.—Flat-topped ridge of small extent rising to about 8 feet above the surroundings and running parallel to the river. Size of pit: 50 yds. × 30 yds. × 2 yds. = 3,000 cu. yds. Average depth of excavation = 6 feet.

Character of material.—Fairly well-graded gravel is exposed in walls from 4 to 6 feet high. The material is horizontally stratified and packed hard. Flat water-worn pebbles, largely composed of Chazy sandstone. Boulders occur in lesser quantity than in the Kirby pit, but stones over 3 inches are more frequently encountered than in the other pits of the locality.

The overburden consists of 1 to 1½ feet of loamy and somewhat weathered gravel.

Two samples were taken, Nos. 6 and 7. They respectively represent the coarsest and finest materials of this occurrence.

The probable amount easily available is over 20,000 cubic yards, with a haul of only a few hundred feet to the main road.

VI. Vankleek Hill pit, 2 miles southwest of the town

Location.—Lot 14, con. VI, West Hawkesbury township.

Description of deposit.—The deposit forms an important sharp ridge trending in a northeast direction across the road between concessions VI and VII. There are two pits opened immediately north of the road, at two different levels. The excavation near the top of the ridge is 100 yds. × 35 yds. × 2 yds. = 7,000 cu. yds. in size, and the pit at the foot of the slope is 50 yds. × 25 yds. × 3 yds. = 3,750 cu. yds. The extent of the area over which there are good gravel surface indications is at least 300 yds. × 100 yds., with an elevation of about 30 feet.

Character of material.—In the upper level pit, a 5-foot wall shows well stratified layers of rounded gravel under 3 inches in size, interbanded with layers 1 to 1½ feet thick of finer material. The amount of boulders over 3 inches is small, but there are streaks of loam in the gravel. Marine shells are also found, but in small amount. Walls are standing up straight.

In the lower pit, coarser material occurs. The percentage of boulders over 6 inches in diameter is, however, low. The gravel lacks fine material under ¼ inch in size, which has apparently been washed away. The pebbles are coated with calcium carbonate. They are mostly limestone. Straight sand is absent in this exposure. Samples of both the coarse and the fine material were collected. Good road gravel can be had from either pit.

The amount available here with only very little stripping is large. It can be safely estimated at over 100,000 cubic yards. See samples Nos. 8 and 9 in Tables II and III.

RUSSELL COUNTY

VII. Devlin pit, Clarence Point

Location.—Pit located on the Ottawa-Point Fortune highway at a point 1 mile south of the town of Clarence.

Description of deposit.—The deposit occupies the triangular area between the main road and the road running south to Clarence Creek. The top of the deposit is only a few feet higher than the road level and the thickness of the gravel is apparently not more than 6 to 8 feet. The amount of material taken out along the road is approximately 1,000 cubic yards.

Character of material.—The exposure consists of a few feet of gravel overlain by a thin overburden. The gravel is of fine texture and interbedded with sand. The amount of weathering ranges from 2 to 2½ feet from the surface, but underneath clean fine concrete gravel is exposed. There are a few big boulders lying in the excavation, but no coarse gravel exposed.

Several thousand cubic yards of fine gravel can be had here without trouble. Chances are that a 6 to 8-foot working face can be developed by the road.

Sample No. 10 was taken.

VIII. Rathwell pits, Bearbrook

Location.—Lot 18, con. IV, Cumberland township. About 1½ mile south of Leonard station.

Description of deposit.—Extensive ridge of sand and gravel running in a southeasterly direction along the Leonard-Bearbrook road.

The deposit, which parallels, and at its northern end is cut by the Leonard-Bearbrook road, forms a prominent ridge, over 30 feet high and a few hundred yards wide for a distance of nearly one-quarter of a mile to the east of the road.

There are two main excavations.

Character of material.—Pit A.—Pit at northern end of deposit, immediately east of road.

Size of excavation, 40 yds. × 30 yds. × 2 to 10 yds. = 7,200 cu. yards.

The main wall is 30 feet high. The overburden consists of 1 to 1½ feet of sandy loam. In the upper 6 to 8 feet stratified fine yellow sand is exposed. It is underlain by interbedded layers 2 to 4 feet thick of sand and gravel. The gravel is rather coarse and contains boulders up to 10 inches in size. In the total height of the exposure, sand occurs in larger proportion than gravel. Large quantities of gravel have apparently been obtained from this pit, but judging by the actual sections exposed along the main wall and in a nearby smaller excavation the material is nearly exhausted. The deposit becomes one of sand.

Samples of the sand and of the gravel were collected, see Nos. 11 and 12, Tables II and III.

Pit B.—Located a short distance south of pit A, above described. This is the more important of the two pits as regards the amount available and the quality of gravel for road purposes.

The pit is opened on the western slope of the ridge, which is quite steep and over 30 feet high at that point. The actual size of the excavation is 40 yds. × 20 yds. × 6 yds. = 4,800 cu. yds. The maximum height of wall developed is 35 feet.

The exposure consists entirely of rather coarse gravel occurring in the form of a huge pocket or a thick lense. Stratification is not apparent and layers of straight sand are absent. The walls stand up straight and the material is apparently packed hard and cementing. The percentage of pebbles between 3 and 10 inches in diameter is large, but boulders over 10 inches are scarce. The depth of weathering and overburden of loamy gravel is only 1 foot. The pebbles are composed mainly of limestone with sandstone and shale. The amount of good road gravel to be had from this deposit is large. It is hard to tell how far back the gravel pocket extends. There may be a sharp passage from gravel into straight sand, as such is seen in the northern pit. However, the amount available may be roughly estimated as several times the quantity already taken out.

The results of analysis of sample No. 13, which represents the average bank run, are given in Tables II and III.

IX. Leonard station, C.P.R. pit

Location.—Immediately north of the C.P.R. line, one-quarter of a mile east of Leonard station.

Description of deposit.—Elongated, rather low, flat-topped ridge lying in a northwest direction. The deposit extends over an area of approximately 1 mile × ¼ mile. The main pit from which ballast material was formerly obtained is now abandoned. It is over 500 yards long and its average width is 60 yards. The height of the banks ranges from 5 to 25 feet. Just west of the entrance to this main pit, near the railway tracks, there are a few small excavations from which several hundred cubic yards of gravel were obtained for road work.

Character of material.—Part of the banks in the main pit are covered with talus, but the various sections of exposure show that in the northern half of the deposit sand for the most part occurs. In the west bank of the southern half of the excavation good gravel is exposed. It is horizontally

stratified with yellow sand. As a rule the sand is coarse, carrying a low percentage of pebbles. The gravel is bouldery but the amount of large boulders lying in the pit or seen in the bank is comparatively small. The depth of weathering varies from 1 to 3 feet. That portion of the bank where gravel occurs is standing up fairly well but the material in it is not packed very hard.

The composition of the pebbles includes several types of igneous rocks, limestones, sandstones, etc., and the proportion in which these various constituents are found varies a great deal from place to place in the deposit. The percentage of weathered and soft rocks is also large. The pebbles are partly coated with calcium carbonate and sand grains adhere to them.

The total amount of sand and gravel available in this deposit is approximately over 200,000 cu. yds., but the chances are that the quantity of suitable road gravel will not be over 20 per cent of this yardage. Two samples (Nos. 14 and 15) of gravel and one sample (No. 16) of the underlying sand, were taken from two different places a few hundred feet apart along the west bank near the entrance of the pit. The results of the analysis will be found in Tables II and III.

X. Walsh pit, Leonard station

Location.—A short distance east of the C.P.R. station. The pit is opened beside the road just north of and parallel to the railway. This is on the southwestern margin of the deposit described in Section IX.

Character of material.—Good road gravel of medium size. Some of it has been used for road work. The amount taken out is small, the excavation being only 35 yds. × 20 yds. × 2 yds. = 1,400 cubic yards.

In a 6-foot wall, interbedded layers of gravel and coarse sand are exposed, both materials occurring in about equal amount. The overburden is thin and the amount of boulders present is small. More material is to be had from this pit.

One sample was taken. See No. 17, Tables II and III.

XI. G. C. Hayes' pit, Cumberland

Location.—Lot A, concession IV, Cumberland township, 2 miles south of the town of Cumberland.

Description of deposit.—The deposit, which is approximately 400 yds. by 100 yds. in extent, overlies the bed-rock. It lies in the western slope of an extensive ridge of limestone trending northeasterly. The dimensions of the pit are 75 yds. × 75 yds. × 2 yds. = 11,250 cu. yds. It is doubtful whether the gravel reaches a depth of more than 6 to 8 feet.

Character of material.—This is an erosion limestone gravel containing very little fine. The sand part is composed of limestone particles and lumps carrying earthy material of a dark colour. The limestone pebbles are well rounded and white coated with calcium carbonate. The material is not packed hard, and there are streaks of gravel from which all of the fine material has been washed away. Clay and loam occur mixed with the fine, but in moderate quantity. Boulders over 6 inches in diameter are very few. The overburden consists of 10 inches to 1 foot of weathered gravel and loam. The amount of material still available is small.

Four samples were taken. Three (Nos. 18, 19, and 20) from the west wall of the pit at various depths from the surface, and one (No. 21) from the northern part of the excavation. The results of analyses are given in Tables II and III.

Results of Tests on Gravel Samples

The results of granulometric analyses of 21 samples of sand and gravel collected are given in Table II. Table III contains information on the character of the material, on the proportion of gravel (material over $\frac{1}{4}$ inch) to sand (material under $\frac{1}{4}$ inch), on the composition of pebbles as to their durability, and on the amount of impurities present.

It may be said that most of the gravels examined are clean and unweathered. Even those carrying more than 20 and less than 50 per cent of soft material will generally be suitable for light country traffic such as exists on county roads in this part of the country.

TABLE II

Results of Tests on Gravel Samples, Russell and Prescott Counties, Ontario

Granulometric Analysis

Sample No.	Location	Gravel: Per cent retained on screens							Sand: Per cent retained on sieves						% passing 200	Remarks
		2½″	2″	1½″	1″	¾″	½″	¼″	8	14	28	48	100	200		
1.....	Bélanger pit. Point Fortune.....	....	....	....	2	4	10	84	16	10	37	25	9	3	2	From layer B.
2.....	Bélanger pit. Point Fortune.....	43	14	15	9	5	5	8	10	13	38	26	8	3	2	From layer C.
3.....	Cut along road. Point Fortune....	49	4	10	7	8	12	11	16	14	28	22	11	6	3	
4.....	Cousineau pit. W. of Point Fortune.	21	25	18	12	9	10	5	4	2	3	14	58	13	6	
5.....	Kirby pit. Chute a Blondeau.....	35	18	12	10	6	8	11	15	19	27	17	8	5	9	
6.....	Dandy pit. Little Rideau.....	40	7	6	18	4	9	15	18	11	21	29	10	4	6	
7.....	Dandy pit. Little Rideau.....	....	10	21	18	7	11	31	16	16	26	25	10	4	3	
8.....	Vankleek Hill pit. 2 m. S.W. of town.....	51	14	12	9	3	4	6	28	17	18	13	7	4	13	
9.....	Vankleek Hill pit. 2 m. S.W. of town.....	....	2	10	21	15	21	32	18	23	28	17	3	3	7	
10.....	Devlin pit. Clarence Point.....	....	....	6	10	12	20	52	16	16	20	30	12	3	3	
11.....	Rathwell pit. Bearbrook.....	....	....	....	....	....	....	....	....	....	1	2	51	40	6	From sand pit.
12.....	Rathwell pit. Bearbrook.....	45	8	11	14	6	6	10	11	21	37	18	5	3	5	Pit A.
13.....	Rathwell pit. Bearbrook.....	41	22	7	10	7	6	8	30	18	14	12	9	7	10	Pit B.
14.....	Leonard Sta. C.P.R. pit.....	32	9	19	18	7	6	10	8	28	38	13	4	2	6	Upper part of wall.
15.....	Leonard Sta. C.P.R. pit.....	9	....	2	16	19	21	32	9	11	22	28	21	7	3	Lower part of wall.
16.....	Leonard Sta. C.P.R. pit.....	....	....	....	28	....	32	40	3	17	44	24	6	2	4	Underlying sand.
17.....	Walsh pit. Leonard Sta.....	5	5	15	22	13	20	21	11	12	25	32	11	4	5	
18.....	Hayes pit. Cumberland.....	....	11	14	35	15	17	8	10	9	13	15	17	17	19	Southwest bank. 1 ft. from surface.
19.....	Hayes pit. Cumberland.....	....	9	23	43	9	10	6	19	11	13	16	13	11	17	West bank, 1½ ft. from surface.
20.....	Hayes pit. Cumberland.....	....	9	18	23	17	16	17	16	20	30	22	7	2	3	West bank, 4 ft. from surface.
21.....	Hayes pit. Cumberland.....	....	....	10	7	9	27	47	40	15	13	9	7	6	10	Northern wall.

TABLE III

Results of Tests on Gravel Samples, Russell and Prescott Counties, Ontario

Character of Gravel

Sample No.	Proportion of gravel to sand per cent of		Composition of pebbles per cent of			Colour test	Remarks
	Gravel	Sand	Durable	Intermediate	Soft		
1.....	15	85	33	46	21	100	Very fine gravel. Good concrete aggregate. Per cent of material between ¼-inch and 1 inch too low, and proportion of sand too large for top course gravel. Binder content also very low.
2.....	75	25	12	39	49	100	Gravel is coarse; sand is fine. Too large per cent of material over 1 inch. High per cent of soft material.
3.....	65	35	33	38	29	0	Fairly well graded gravel. Contains pebbles over 2½ inches to be raked off if used in surfacing.
4.....	75	25	30	43	27	100	Gravel rather coarse; sand too fine.
5.....	65	35	9	33	58	200	Coarse gravel. Well graded sand. High per cent of soft material. Moderate amount of impurities.
6.....	85	15	15	61	24	100	Small proportion of sand. Large per cent of pebbles over 2½ inches.
7.....	55	45	15	70	15	200	Well graded for surfacing purposes.
8.....	90	10	15	79	6	200	Lacks fine material. Sand well graded and containing proper amount of clay binder, but its proportion to gravel content is low.
9.....	60	40	9	80	11	200	Properly graded for road surfacing. Enough binder.
10.....	35	65	10	80	10	100	Well graded, but high per cent of sand.
11.....	.....	100	.....	.....	.....	100	Too fine sand for concrete.
12.....	50	50	12	54	34	0	Fairly well graded road gravel. Clean.
13.....	85	15	9	67	24	50	A little low in sand, but suitable for use in the base course of gravel roads.
14.....	45	55	30	27	43	0	Sand too uniform and in too large proportion for surfacing. Good concrete aggregate.
15.....	30	70	10	67	23	50	Better grading than No. 14 for road work, but too much sand.
16.....	5	95	26	53	21	50	Poorly graded sand for concrete.
17.....	55	45	10	60	30	0	Well graded road gravel. Suitable for use in top course.
18.....	90	10	.....	80	20	500	Well graded limestone gravel. The per cent of sand is low, but on account of a high content of clay and organic matter, this material should bind well.
19.....	90	10	.....	75	25	500	
20.....	55	45	.....	95	5	500	
21.....	85	15	.....	90	10	500	

TABLE IV

Results of Tests on Gravel Samples, Russell and Prescott Counties, Ontario

Sand Mortar 1:3 mix

Sample No.	Locality	Fineness modulus	% clay and silt	Colour test	% water used	Per cent of strength relative to that of Standard Ottawa sand (1)				Remarks
						Tensile strength		Compressive strength		
						7 days	28 days	7 days	28 days	
1	Bélanger pit, Point Fortune.....	2·84	5	100	13	110	126	127	124	Good.
2	" "	2·76	5	100	11	141	148	166	145	Good.
3	" "	2·75	8	0	12	129	140	168	164	Good
4	Cousineau pit, W. of Point Fortune.	1·23	15	100	16	88	102	65	79	Too fine for concrete.
5	Kirby pit, Chute à Blondeau.....	2·74	10	200	14	104	111	130	126	Fairly good.
6	Dandy pit, Little Rideau.....	2·70	8	100	13	125	136	141	136	Fairly good
7	" "	2·83	10	200	17	95	106	93	70	Low, probably due to impurities.
9	2 miles S.W. of Vankleek Hill.....	3·06	10	200	11	140	146	171	155	Very good, though colour test shows presence of impurities.
10	Devlin pit, Clarence Point.....	2·76	5	100	12	79	110	122	101	Rather low.
11	Rathwell pit, Bearbrook.....	0·58	6	100	20	47	79	44	55	Unsuitable for concrete.
12	" "	2·91	5	0	12	127	166	138	144	Very good.
13	" "	2·97	15	50	16	119	121	145	119	Good.
14	C.P.R. pit, Leonard station.....	2·99	6	0	12	129	142	130	134	Good.
15	" "	2·29	3	50	12	93	147	128	115	Rather low, due to fineness and un-uniform grading.
16	" "	2·69	4	50	12	104	91	93	105	Rather low, due to fineness and un-uniform grading.
17	Walsh pit, Leonard station.....	2·53	5	0	13	108	115	120	137	Rather low, due to fineness and un-uniform grading.

(1) Ottawa standard sand, strength of 1 : 3 mortar:

Amount of water used—10%

Tensile	7 days—	234	lbs. per sq. in.
"	28 days—	282	lbs. per sq. in.
Compressive	7 days—	1,510	lbs. per sq. in.
"	28 days—	2,600	lbs. per sq. in.

III

ROAD MATERIALS SURVEY IN ROCKY MOUNTAINS PARK

In compliance with a suggestion from the Department of the Interior, investigations of road materials were undertaken in the Rocky Mountains Park in 1919,[1] and continued during the summer of 1921.

These investigations were carried on in co-operation with the Parks Branch highway engineers.

The work consisted in locating, examining, sampling, testing and reporting on the comparative road-making qualities of the material occurring along various park roads under construction, so that the most suitable material may be chosen for their surfacing.

Up to date, materials along the Banff-Lake Louise road, a distance of 35 miles, were examined and investigations were also carried on along the Banff-Windermere road.

When the examination of materials along these roads was commenced in 1919 by Dr. K. A. Clark,[1] the road grade on the Banff-Lake Louise road ended a few miles west of Castle, and on the Banff-Windermere road only that portion of the proposed road from Castle southward to the Vermilion summit was opened, a distance of about 17 miles. Since then, construction work has been completed between Castle and Lake Louise and the road opened for traffic during the early part of the 1921 season. The completion of this highway gives to motorists from Calgary and Banff access to the beautiful Lake Louise and Lake Moraine districts.

The road material survey along the final 11 miles of this important highway was resumed and completed this year.

The grading of the Castle-Windermere road is not yet completed, but construction work is proceeding, and before the end of next summer the road will likely be opened to tourist traffic. This road, which forms a link connecting the Bow valley road with the Columbia river highway, will open up a loop route through the Canadian Rockies connecting our national parks with the United States park-to-park system of highways. With its completion, it will be possible for tourists to motor from the prairies via Calgary and Banff, to Windermere, B.C., in the Columbia valley, to Lethbridge, and back to the plains. American motorists will also be able to travel from the national park-to-park highway and the United States Glacier park via Macleod and Calgary through our Canadian Rockies via Banff, Windermere, Cranbrook and back to the United States, thence via the Sunset and Pacific coast highways, to California.

This road undoubtedly means much to the future of the Rocky Mountains park, as it will render its attractive scenic beauties accessible to motor traffic from both east and west.

The distance from Castle to the Columbia valley is about 75 miles. From Castle the road leads to the Vermilion summit, thence to the Kootenay valley, and through the Sinclair pass to the Columbia valley.

An examination of the material occurring along this highway was carried on last summer. This survey included two sections of the road—at the north end, the Vermilion river division, that is from Marble canyon, just south of the Vermilion summit, southward along the Vermilion river

[1] Mines Branch, Summary Report, 1910, pages 139-147.

to Hawk creek, a distance of about 10 miles; at the southern end of the road, the Sinclair Pass-Kootenay Valley division, that is from the Columbia river highway northward to near where the road crosses the Kootenay river to enter a pass to the Vermilion valley. The distance covered at that end of the road is approximately 21 miles. There still remain about 25 miles of this road to be surveyed for road materials. With this work accomplished, complete information regarding the available road materials for this road will be at the disposal of the engineers and contractors in charge of its surfacing. In this examination, fairly suitable material has been located at intervals within economical hauling distance.

SOURCES OF SUPPLY OF ROAD MATERIAL

In mountainous country such as Rocky Mountains Park the roads naturally follow, as much as possible, the courses of the valleys, or are built on the side of the foothills.

These valleys, as a rule, are filled with drift material, and sand and gravel is of general occurrence along the stream channels. The foothills are generally made up of unconsolidated materials which vary from sand and gravel to stony soils, boulder clay and clay. Rock outcrops are seldom met with along the roads in the flats, but they are plentiful in the passes where extensive cutting is often involved.

The Banff-Lake Louise road, which runs up the Bow valley from Banff, keeping to the right hand side of the river, is constructed partly in the valley flat, and partly in side cuts along the foothills.

The Castle-Windermere road, owing to the nature of the country it traverses, is for its greater part built on hillsides which necessitated very heavy work, particularly through the Vermilion and the Sinclair passes.

Because of the nature of the country, materials for the surfacing of these roads must be immediately accessible from the right of way. The different kinds of unconsolidated material and rock employed in highway construction in mountainous country are rarely transported for long distances, local sources of supply being usually drawn upon. The problem was to locate the best material exposed from place to place along the right of way, rather than to seek the most desirable type of material which might occur at a distance from it. It therefore devolves upon the engineer to carefully examine these local sources with reference to the quality, quantity, and accessibility of the materials available. Generally, at short intervals these roads cut through shoulders of gravelly material, which in some cases, is fairly satisfactory for surfacing purposes. Such material includes stone, sand and gravel, and boulder clay.

ROAD STONE

Rock outcrops along the Banff-Lake Louise road from Castle to Laggan are few, the only occurrences of extent being one of grey slate and one of much weathered rusty sericite schist, a short distance east of the railroad crossing. Both rocks are unsuitable for road work.

Crushed stone, however, if desired for surfacing, could be obtained from boulders which are to be found in sufficient quantity along most parts of the road. These boulders occur in large quantities in the boulder

clay, and, as a rule, they are sound and of such size as to permit handling in a small crusher without preliminary breaking. The aggregate is mainly composed of hard and tough limestone and quartzite.

On the Castle-Windermere road, any amount of stone for road work could be quarried from rock cuts along the right of way throughout the Vermilion and the Sinclair passes, but on the other sections of this road that were visited there are very few opportunities for quarrying. There, again, boulders from bouldery soils and coarse gravels could be used for the production of crushed stone. The bulk of this material along the Vermilion and Kootenay rivers and in the Sinclair pass is limestone and dolomite.

Among the more important outcrops from which stone could be obtained are the following:—

Sinclair Canyon

Cliff-over 75 feet high of fine-grained, buff-coloured limestone, partly fresh, partly weathered. The amount easily available is unlimited.

About one-quarter mile north of Sinclair Canyon

On either side of the pass, which is very narrow at this point, there are outcrops of uneven textured, greyish limestone, containing veinlets of calcite. Some of this stone is fairly tough but much of it is weathered and soft. Large quantities of loose material composed of rock fragments are to be found in talus just south of the above mentioned rock outcrops. The stone resembles in character that of the bed-rock but is much more weathered. Most of these fragments are of small size and could be used in road foundation work, or even in the lower course of macadam without being run through a crusher. One sample was taken and tested. See Table V. The ability of this material to resist wear is comparable to that usually shown by medium soft limestone.

Immediately north of Radium Hot Springs

Buff-coloured limestone or dolomite is exposed in rock cuts along the road. Material for crushed stone could be easily obtained.

Red Bluff, about one-quarter mile north of Radium Hot Springs

Brick red to pink dolomite forming huge cliff. The stone is iron stained and contains geodes and veinlets of secondary calcite. It is somewhat weathered and rather soft.

Very large quantities of this material are readily available from a talus over 50 feet high just south of the cliff. The disintegrated rock constitutes an aggregate which is fairly well graded from dust to gravel sizes and could be used as gravel for road surfacing purposes. The results of analysis of one sample collected are given in Tables XI and XII. See No. 56.

Abrasion and cementing tests on this stone have shown that it is not resistant to wear but that it binds well. The road surface along the talus where this material occurs is well compacted and apparently wears uniformly, but it is dusty in dry weather.

South of bridge, three-quarter mile north of Radium Hot Springs

Big rock cut, on road. Fresh, dark coloured limestone, very fine-grained, with numerous irregular shaly partings and veinlets of calcite. Breaks very irregularly with sharp edges. This limestone is harder and wears more slowly than the red dolomite and the lighter coloured limestones occurring in the southern portion of the pass. Large quantities of it could be obtained from debris or quarried without trouble. One sample was taken and tested. See Table V.

Similar limestone is well exposed in rock cuts in several instances, at short intervals along the road, between mile 3¼ and mile 5¼.

From Summit Lake to Kootenay Valley

Rock outcrops are frequently encountered along the right of way, but they cannot be regarded as possible sources of road material. The exposures are either of black, rusty, thinly foliated shales, or whitish talcous schist, both classes of rock being unsuitable for road work.

GRAVEL

Gravel is of pretty general occurrence along the Banff-Lake Louise road, and in the Sinclair pass and Kootenay sections of the Banff-Windermere road.

From Castle to Laggan the gravel, as a rule, occurs in the valley flats, but how thick the deposits are cannot be told without boring or test-pitting. Insofar as could be observed in borrow pits made in connection with the grading of the road, these surface gravel deposits are only of small thickness and are underlaid by the clays. They carry in most cases loam, clay or silt, but in rather small proportion.

The gravels to be found along the Vermilion section of the Banff-Windermere road occur in pockets or in strata through the boulder clay. The deposits often carry stratified layers, lenses or lumps of clay. The clay and silt content of these gravels is therefore generally much higher than that of the gravels in the Bow valley.

In the Sinclair pass and along the Kootenay valley the amount of fine material passing 200 mesh carried by the gravels is also quite high, but of a more silty nature.

In carrying on this road material survey nearly all the gravel occurrences encountered were sampled. These samples number over 40. They were examined and tested as to their suitability as road surfacing material or as concrete aggregate.

BOULDER CLAY

Along the above-mentioned sections of road boulder clay deposits in which the proportion of stone content to that of clay is high, were noted and sampled. This material is seldom just what is required to provide a really suitable surfacing material. It is often too coarse or unevenly graded, deficient in fine material, or full of silt and clay. However, some of the deposits examined contain material that might furnish a fairly good aggregate. Similar material was used in road surfacing and fairly good results were obtained. A glance at the results of analysis which are recorded in Tables VII to XII will show that some of the boulder clay samples compare well with the gravels of the same locality.

For the purpose of comparison, values obtained in the granulometric analysis of 60 samples of gravel, 20 samples of boulder clay and a number of samples taken from gravel road surfaces in Rocky Mountains Park, have been condensed in a table by Dr. K. A. Clark.[1] His figures are partially reproduced in Table VI, together with the limits shown by the screen analysis of the samples collected during 1921.

The limits for analysis of samples of good gravel road surfaces can be taken as reference limits in the comparison of the various materials as to their suitability for road surfacing purposes. The road making quality of an aggregate such as gravel or boulder clay is largely dependent on the following factors, fineness modulus, per cent of coarse, per cent of cement (material passing 200 mesh), and physical properties of cement. With reference to these factors, the difference in the proportion of material passing 200 mesh is probably the most important when comparing gravels or boulder clays, as it is this fine material that in the case of a gravel road makes the difference between a compacted and a loose surface, while in boulder clay roads it makes the surface hard when dry, but renders it muddy and slippery when wet.

[1] Mines Branch, Summary Report, 1919, Road Materials in Rocky Mountains Park, page 145.

TABLE V

Results of Physical Tests upon Bed-rock. Rocky Mountains Park

Locality	Rock type	Physical properties							Remarks
		% wear	French coef. of wear	Toughness	Hardness	Specific gravity	Absorption in lbs. per cu. ft.	Cementing value	
¼ mile N. of Sinclair canyon (Banff-Windermere road).	Limestone	4·7	8·5	Low	.....	2·76	0·5	.....	Uneven textured. Rock fragments. Talus material. Partly weathered.
North of Radium Hot Springs.	Dolomite	5·4	7·3	Low	.....	2·65	1·2	77	Brick red to pink altered rock. Forms big cliff. Fairly well graded aggregate to be had from talus.
About ¾ mile north of Radium Hot Springs.	Limestone	4·10	9·7	Medium	16	2·71	0·3	24	Fresh, dark coloured fine-grained limestone with veinlets of calcite and irregular shaly partings.

TABLE VI

Summary of the Results of Granulometric analyses of samples from surfaces of Gravel Roads and of samples from Gravel and Boulder Clay Deposits

Location	Number of samples	Per cent of gravel (material over ¼ inch,) in total aggregate		Per cent passing 200 mesh in material under ¼ inch.	
		Limits	Average	Limits	Average
1919 survey—					
Gravel from gravel road surfaces.....	7	35 to 60	.....	25 to 35	
Gravel from deposits along various park roads.....	60	40 to 75	.....	5 to 30	
Boulder clay deposits along various park roads.....	20	20 to 60	.....	25 to 50	
1921 survey—					
Gravel along Castle-Laggan road.....	21	10 to 85	50	2 to 45	12
Boulder clay Castle-Laggan road.....	8	35 to 60	45	25 to 55	35
Gravel along Vermilion section of the Windermere road.....	8	35 to 80	55	8 to 55	25
Boulder clay along Vermilion section of the Windermere road.....	12	30 to 65	50	25 to 65	45
Gravel along Sinclair pass-Kootenay section of the Windermere road.....	13	40 to 80	60	10 to 35	25
Boulder clay along Sinclair pass-Kootenay section of the Windermere road..	11	35 to 70	55	25 to 55	35

TABLE VII

Results of Physical Tests on Gravel and Boulder Clay Samples, Rocky Mountains Park

ALONG BANFF—LAKE LOUISE ROAD (Eldon-Laggan Division)

Sample No.	Location of deposit (chainage)	Type of material	Granulometric analysis																Remarks
			Proportion of gravel to sand		Gravel — Per cent retained on screens							Sand — Per cent retained on sieves						% passing 200	
			Per cent grav.	Per cent sand	2½″	2″	1½″	1″	¾″	½″	¼″	8	14	28	48	100	200		
1	340	Gravel	60	40	0	9	8	32	13	16	22	14	14	21	29	15	5	2	Fairly well graded for road work
2	340–345	Boulder clay	.....	100	.....	.....	.....	.....	.....	.....	.....	9	10	10	12	18	21	20	Yields a firm foundation. Needs little gravel on top.
3	360–365	Gravel	60	40	0	4	3	20	12	25	36	32	15	11	15	15	7	5	Well graded.
4	400–405	Gravel	65	35	44	10	9	15	6	6	10	14	18	19	18	14	9	8	Not as good grading as above, but contains more binder.
5	465	Gravel	45	55	0	12	24	19	9	16	20	8	12	18	22	18	11	11	High clay content.
6	465	Gravel	85	15	0	25	33	26	6	6	4	6	8	11	14	14	17	30	Very high clay content.
7	466	Gravel	57	43	0	9	17	24	13	16	21	17	12	14	24	23	7	3	Could be used for concrete.
8	500	Gravel	60	40	25	7	9	15	8	13	23	21	22	29	19	6	2	1	Lacks binder, to be used as surfacing material.
9	500	Boulder clay	45	55	11	4	20	16	11	15	23	14	10	11	14	22	19	10	Will pack well in a road bed but surface will be dusty.
10	500	Boulder clay	58	42	25	20	15	13	7	8	12	8	13	15	19	24	14	7	
11	525	Gravel	70	30	22	8	20	15	10	10	15	21	11	19	33	10	3	3	Suitable for concrete.
12	550	Gravel	73	27	0	0	7	26	13	22	32	32	6	2	6	32	14	8	Loamy.
13	558–560	Gravel	58	42	0	16	9	19	14	16	26	15	16	26	23	7	5	8	
14	585–590	Gravel	–	–	21	14	11	21	12	11	10	36	22	22	12	3	2	3	Fairly well graded, lacks binder
15	595	Gravel	40	60	8	25	8	13	6	12	28	13	19	41	19	3	2	3	Lacks binder. Suitable for concrete.
16	615	Boulder clay	60	40	0	0	17	23	24	14	22	21	16	15	15	15	10	8	Well graded aggregate.
17	620–623	Boulder clay	55	45	0	15	19	24	11	12	19	17	12	14	18	19	13	7	Well graded aggregate.
18	640	Clayey gravel	30	70	0	0	3	10	10	24	53	22	17	20	23	10	4	4	Good clayey fine gravel.
19	660	Boulder clay	55	45	11	9	15	13	16	16	20	14	13	16	22	19	11	5	Well graded aggregate. Packs well.
20	707–714	Gravel	70	30	0	12	23	29	8	12	16	23	16	18	23	12	5	3	Well graded but lacks binder.
21	714–717	Boulder clay	43	57	0	0	4	13	9	18	56	30	18	14	12	12	9	5	Good surfacing material.
22	714–717	Gravel	58	42	4	19	14	20	12	13	18	12	11	27	35	10	3	2	Can be used for concrete purposes
23	786	Gravel	60	40	5	6	14	19	11	17	28	18	24	30	17	7	3	1	Well graded, lacks binder

24	800–805	Boulder clay..	35	65	0	0	15	11	13	23	38	17	13	15	17	15	10	13	Sample taken from road crust.
25	808–810	Gravel.....	20	80	0	0	0	4	7	22	67	13	8	15	31	23	8	2	Suitable for concrete work.
26	815	Gravel.....	55	45	11	4	12	16	9	13	35	23	22	15	13	14	10	3	Well graded. Carries enough binder to be used in surfacing work.
27	830	Gravel.....	42	58	0	0	2	21	10	22	45	23	11	11	23	25	5	2	
28	830	Gravel.....	70	30	0	12	32	35	13	6	2	2	1	4	35	41	11	6	Poorly graded. Too fine sand. Silty.
29	845–851	Gravel.....	10	90	0	0	7	11	9	20	53	8	19	31	29	7	2	4	Too sandy for road surfacing. Could be used in concrete.
30	867	Gravel.....	12	88	0	0	7	22	8	18	45	10	21	42	22	3	1	1	

TABLE VIII

Results of Physical Tests on Gravel and Boulder Clay Samples, Rocky Mountains Park

ALONG BANFF-LAKE LOUISE ROAD (Eldon-Laggan Division)

Sample No.	Type of material	Character of material						Amount available	Remarks
		Composition of pebbles			Remarks	Shape of pebbles	Clay and silt		
		Durable	Intermediate	Soft					
		%	%	%			%		
1	Gravel	45	45	10	Largely quartzite	Subangular	10	Small	Loamy.
2	Boulder clay				Limestone and dolomite		55		
3	Gravel	0	75	25	Dolomite predominates. Some sandstone and shale.	Rounded	9	Small.	
4	Gravel	30	50	20	Largely metamorphosed limestone.	Subangular	29	Probably large	Loamy and clayey, loose. Ridge.
5	Gravel	35	50	15	Largely quartzite with limestone and dolomite, partly altered.	Partly round, partly angular	48	"	" " "
6	Gravel	45	45	10			58	"	" " "
7	Gravel	45	50	5	" "	"	7	Small	Clean, in flat.
8	Gravel	40	30	30	" "	Mostly angular.	4	Fairly large	Loose sandy to bouldery, in flat.
9	Boulder clay	10	80	10	Largely metamorphosed limestone or dolomite.	Rounded to angular.	26	Very large	Well compacted. Hillock at Baker creek.
10	Boulder clay	40	45	15		"	54	"	
11	Gravel	50	30	20		Angular	6	About 1,500 cu.yd.	In flat.
12	Gravel	30	60	10		Angular	31	Very small	Loamy. Packs well.
13	Gravel	30	55	15	Pebbles coated with $CaCO_3$.	"	9	Over 5,000 cu.yds.	Bouldery deposit.
14	Gravel	40	40	20	D.—Quartzite I.—Marble and sandstone.	"	8	1,000 cu. yds	Bouldery. Deficient in binder.
15	Gravel	30	55	15	D.—Quartzite I.—Limestone.	Rounded to subangular.	4	Small	Loose washed gravel in flat.
16	Boulder clay	35	50	15	"	"	23		Loose, loamy. Ridge.
17	Boulder clay	35	50	15	"	"	27	Small	In road cuts. Well compacted opposite Temple Sta.
18	Clayey gravel	20	70	10	I.—Largely marble	"	18	Probably exhausted.	Was used with success last summer in road surfacing work.

19	Boulder clay..	40	50	10	"	.....	31	" ..	" " "
20	Gravel.....	60	35	5	D.—Largely quartzite.....	Fairly angular	9	Large.....	Loose, loamy and bouldery.
21	Boulder clay..	35	60	5	$CaCO_3$ coating.....	Fairly angular	25	Fairly large.....	Can be readily obtained from big cut along road.
22	Gravel.....	25	65	10	.....	Fairly angular	6	"	
23	Gravel.....	35	20	45	High content of soft flat shales.	Many flat	4	About 1,000 cu.yd.	
24	Boulder clay..	20	35	45	" " ..	pebbles. .. "	31	.....	Sample taken from road crust.
25	Gravel.....	30	35	35	" " ..	" ..	10	Small.....	Very sandy.
26	Gravel.....	30	45	25	I.—Mostly limestone and sandstone.	Largely angular.	13	700 cu. yds.....	
27	Gravel.....	60	35	5	D.—Quartzite..... I.—Limestone and dolomite. S.—Shales.	Subangular...	5	Large.....	Clean, loose sand and gravel.
28	Gravel.....	50	40	10	"	Subangular....	12	Large.....	Silty.
29	Gravel.....	45	35	20	"	Rather angular.	6	Fairly large.....	Turbidity in color test due to clay content. Mortar tested good.
30	Gravel.....	40	40	20	"	" ..	2	Large.....	Clean sand for concrete.

TABLE IX

Results of Physical Tests on Gravel and Boulder Clay Samples, Rocky Mountains Park

ALONG BANFF-WINDERMERE ROAD (Vermilion River Division)

Sample No.	Location of deposit (chainage)	Type of material	Granulometric analysis															Remarks
			Proportion of gravel to sand		GRAVEL							SAND						
					Per cent retained on screens							Per cent retained on sieves						% passing 200	
			Per cent grav.	Per cent sand	2½″	2″	1½″	1″	¾″	½″	¼″	8	14	28	48	100	200		
31	253	Boulder clay...	68	32	13	8	25	18	9	12	15	18	12	12	12	12	12	22	Fairly well graded for surfacing purposes. Opposite Tokkum creek.
32	.....	Boulder clay..	50	50	12	10	13	16	11	16	22	18	8	7	8	14	17	28	North of gate.
33	290–295	Boulder clay..	50	50	0	6	5	25	14	20	30	23	16	15	13	7	9	17	In flat.
34	67	Boulder clay..	40	60	0	3	12	19	15	22	29	15	11	8	9	14	27	16	Packs well on road.
35	80–87	Boulder clay..	33	67	0	0	6	17	18	22	37	16	11	7	7	7	13	39	Very poor road material.
36	116–118	Gravel.....	45	55	4	8	14	15	10	14	35	30	23	18	10	4	2	13	Suitable for road surfacing purposes.
37	145	Sand.....	.....	100	.....	.....	.....	.....	.....	.....	.....	.....	0	2	19	36	20	23	Too fine for concrete.
38	156	Boulder clay..	68	32	35	3	15	10	9	11	17	24	18	15	12	8	8	15	Well graded. Suitable for surfacing purposes.
39	220–225	Boulder clay..	50	50	0	13	18	15	10	17	27	19	17	12	10	7	8	27	Packs well on road. Well graded. Good surfacing material.
40	230	Boulder clay..	45	55	0	8	13	16	13	18	32	19	14	15	13	8	7	24	" "
41	243	Clayey gravel.	80	20	11	6	21	18	10	14	20	24	13	11	14	13	8	17	" "
42	268	Gravel.....	69	31	0	0	2	12	19	25	42	59	16	4	2	2	4	13	Packs and cements well on the road.
43	277	Gravel.....	46	54	0	4	10	20	17	19	30	19	13	10	10	7	7	34	High content of clay and loam.
44	282	Gravel.....	44	56	6	0	22	14	14	19	25	15	11	8	8	9	10	39	High clay content, compacted hard.
45	295–298	Gravel.....	36	64	0	0	17	31	10	17	25	16	17	19	14	10	6	18	Too large proportion of sand. However actual road surface firm and smooth.
46	323–325	Boulder clay..	57	43	4	23	27	15	8	9	14	13	10	9	10	12	20	26	Fairly well graded. High clay content.

47	327–328	Boulder clay...	55	45	19	6	9	24	10	12	20	21	16	13	10	6	6	28	Better material than above.
48	333	Boulder clay..	44	56	0	0	15	19	12	21	33	20	14	12	12	11	7	24	Fairly good material.
49	345	Boulder clay..	45	55	0	21	14	11	12	16	26	20	12	11	12	10	20	15	" "
50	432	Gravel.....	70	30	12	7	12	20	15	15	19	28	17	15	12	8	5	15	Well graded loamy gravel.
51	442	Gravel.....	62	38	0	8	22	18	15	16	21	23	25	29	13	14	2	4	Lacks binder to be used as surfacing material. Clean enough for concrete.

TABLE X

Results of Physical Tests on Gravel and Boulder Clay Samples, Rocky Mountains Park

ALONG BANFF-WINDERMERE ROAD (Vermilion River Division)

Sample No.	Type of material	Character of material						Amount available	Remarks
		Composition of pebbles				Shape of pebbles	% clay and silt		
		Durable %	Intermediate %	Soft %	Remarks				
31	Boulder clay....	5	55	40	Marble, limestone and shaly limestone.	Subangular and flat	40	Small	
32	Boulder clay....	0	95	5	Largely marble.....	Angular	45	Small	
33	Boulder clay....	5	50	45	Largely marble.....	Subangular	34	Very small	
34	Boulder clay....	.....	20	80	Very high % soft calcareous shales and weathered stone.	Flat	55	Fairly large	High in clay content.
35	Boulder clay....	5	15	80	" "	Flat	64	.....	Too high clay content.
36	Gravel.....	5	30	65	" "	Flat	15	Small	Fairly clean concrete aggregate, but high clay content.
37	Sand.....	.....	.....	.....	.....	.....	33	Small	Too silty.
38	Boulder clay....	5	25	70	" "	Flat	26	900 cu. yds.	About proper clay content.
39	Boulder clay....	0	35	65	" "	Flat	38	A few hundred cu. yds.	Loose, loamy material.
40	Boulder clay....	10	65	25	Largely marble and limestone.	Subangular	36	Large	Well compacted material in bank and on road surface.
41	Clayey gravel...	5	60	35	" "	Subangular	28	800 cu. yds.	Road surface a little loose.
42	Gravel.....	5	70	25	" "	Subangular	17	A few hundred cu. yds.	Proper amount of binder.
43	Gravel.....	5	40	55	Largely limestone, shaly limestone and weathered shales.	Angular and flat	50	Small	Very loamy and clayey gravel.
44	Gravel.....	0	35	65	" "	" "	57	Small	" "
45	Gravel.....	5	65	30	" "	" "	21	Small	Too high clay content to be used for concrete.
46	Boulder clay....	0	30	70	" "	Subangular and flat	66	.....	High clay content. Boulders.
47	Boulder clay....	0	35	65	" "	" "	42	Over 1,000 cu. yds.	Well compacted on road.
48	Boulder clay....	0	45	55	" "	" "	48	Fairly large	Very few boulders.
49	Boulder clay....	0	50	50	" "	" "	57	Rather small	Loamy.
50	Gravel.....	5	35	60	" "	" "	20	Probably large	Contains just enough binder.
51	Gravel.....	0	55	45	" "	" "	28	Fairly large	Rather clean concrete aggregate

TABLE XI

Results of Physical Tests on Gravel and Boulder Clay Samples, Rocky Mountains Park

ALONG BANFF-WINDERMERE ROAD (Sinclair Pass—Kootenay Valley Division)

Sample No.	Location of deposit (chainage)	Type of material	Granulometric Analysis															Remarks
			Proportion of gravel to sand		Gravel							Sand						
					Per cent retained on screens							Per cent retained ons ieves						
			Per cent grav.	Per cent sand	2½″	2″	1½″	1″	¾″	½″	¼″	8	14	28	48	100	200	% passing 200	
	Mileage from Columbia Valley road																	
52	½ mile	Boulder clay	53	47	7	5	6	13	14	22	33	21	13	11	13	14	13	15	Well graded, suitable for surfacing purposes.
53	½ "	Gravel	50	50	0	0	0	13	18	23	46	32	21	16	13	10	6	2	Good concrete aggregate.
54	¾ "	Boulder clay	50	50	0	0	10	12	14	30	34	16	15	18	17	17	11	6	
55	Just S. of Radium Hot Spring	Boulder clay	65	35	0	15	16	29	10	11	19	29	20	18	12	6	8	7	Material in bank partly coarser than sample.
56	N. of Radium Hot Springs.	Disintegrated rock (talus).	90	10	16	7	25	23	8	11	10	38	14	10	9	8	9	12	Fairly well graded talus material.
57	2½ miles	" "	67	33	19	12	8	12	11	12	26	37	15	9	7	6	8	18	Talus runs coarser than sample.
58	3¼ "	Gravel	47	53	0	8	4	14	15	24	36	26	14	12	13	12	11	12	Sandy fine gravel. Contains enough binder to be used in surfacing work.
59	4 "	Boulder clay	65	35	15	7	10	15	12	17	24	32	15	9	8	9	15	12	Fairly well graded, compacts well on road.
60	4¼ "	" "	38	62	0	0	15	14	10	20	41	23	17	13	11	10	12	14	Per cent over ¼-inch a little low.
61	4¾ to 5 "	" "	48	52	0	0	33	10	11	25	51	33	18	13	9	8	9	10	Good surfacing material.
62	5½ "	Gravel	70	30	13	3	12	20	12	17	23	30	11	8	11	13	16	11	Loamy fine to medium gravel. Good grading.
63	6¼–6½ "	"	70	30	4	0	8	16	12	23	37	47	16	9	6	5	7	10	" " "

TABLE XI—*Con.*

Results of Physical Tests on Gravel and Boulder Clay Samples, Rocky Mountain Park

ALONG BANFF-WINDERMERE ROAD (Sinclair Pass—Kootenay Valley Division)

Sample No.	Location of deposit (chainage)	Type of material	Granulometric analysis																Remarks
			Proportion of gravel to sand		GRAVEL							SAND							
					Per cent retained on screens							Per cent retained on sieves							
			Per cent grav.	Per cent sand	2½″	2″	1½″	1″	¾″	½″	¼″	8	14	28	48	100	200	% passing 200	
64	6½ miles	Gravel	54	46	0	16	3	7	11	20	43	44	24	13	8	4	3	4	Contains enough binder to be used in surfacing work.
65	7½ "	Boulder clay	70	30	0	0	10	23	18	26	23	26	16	13	12	11	11	11	
66	7·8 "	"	52	48	0	2	4	16	15	23	40	21	11	12	12	10	10	24	Fairly well graded stone aggregate. Contains high clay content.
67	8½ "	Drift	47	53	0	2	10	14	18	27	29	37	19	8	4	4	7	21	Loamy material, loose.
68	Summit Lake.	Gravel	75	25	3	4	11	22	16	23	21	29	14	12	12	11	9	13	Loose loamy gravel in flat.
69	9½ miles	"	40	60	0	2	7	21	13	21	36	16	15	18	18	13	11	9	High per cent passing ¼-inch screen. Loamy.
70	10 "	Boulder clay	67	33	0	0	8	17	14	25	36	37	21	12	8	8	8	6	Well graded.
71	10-10¾ "	Gravel	78	22	0	0	2	8	14	30	46	45	11	8	7	7	12	10	
72	10-10¾ "	Boulder clay	60	40	0	5	8	16	16	23	32	27	18	16	13	11	12	3	
73	Chainage 879–867	Clayey gravel	35	65	0	0	2	11	14	25	48	23	19	17	14	9	9	9	Contains high clay content.
74	776	Gravel	52	48	0	0	14	17	12	23	34	25	13	12	12	11	12	15	Contains clayey loam binder. Well graded fine gravel.
75	753	"	55	45	14	4	7	19	16	16	24	23	18	20	15	9	11	4	Well compacted in bank and on road. Good grading.
76	720	"	67	33	0	2	5	18	13	24	38	37	15	8	7	10	11	12	Very good gravel.
77	697–415	"	83	17	5	14	21	24	12	11	13	38	15	12	15	7	4	9	Clean concrete aggregate. Lacks binder for use in road surfacing.

TABLE XII

Results of Physical Tests on Gravel and Boulder Clay Samples, Rocky Mountains Park

ALONG BANFF-WINDERMERE ROAD (Sinclair Pass—Kootenay Valley Division)

Sample No.	Type of material	Character of material						Amount available	Remarks
		Composition of pebbles				Shape of pebbles	% clay and silt		
		Durable %	Intermediate %	Soft %	Remarks				
52	Boulder clay	5	65	30	Mostly fresh limestone coated with calcium carbonate.	Rounded	24	Large	Looks almost like gravel carrying clay.
53	Gravel	5	80	15	" "	"	10	Fairly large.	
54	Boulder clay	0	70	30	" "	"	45	Large.	
55	"	0	75	25		"	28	Small.	
56	Disintegrated rock (talus).	0	20	80	Brick red altered dolomite	Angular	29	Very large	Compacts well but yields a dusty road surface. Will wear fast.
57	"	0	70	30	Largely limestone	"	27		
58	Gravel	0	85	15	" "	Rounded	23	Small.	Too high clay and silt content for concrete.
59	Boulder clay	5	85	10	" with $CaCO_3$	Subangular	36	At least 2,500 cu. yds.	
60	"	0	75	25	Largely limestone	"	36	1,000 cu. yds.	
61	"	0	90	10	" "	"	26	Large	Good surfacing material.
62	Gravel	0	90	10	" "	"	31	Very small	Loamy.
63	Gravel	0	80	20			26	Rather small	Loamy loose gravel.
64	"	0	90	10			16	Large	Packed hard in bank. Clay and silt content high for concrete.
65	Boulder clay	0	60	40	Limestone and weathered shales.	Angular	47	Small	Rather poor material.
66	"	0	90	10			55	Fairly large	Very high clay content.
67	Drift	0	60	40	Limestone and shales	"	28	Very small	Loam.
68	Gravel	5	20	75	Weathered shale	Rounded to flat.	27	Small	Loam. Ought to bind well.
69	"	0	40	60	" "	"	34	Small	Loamy.
70	Boulder clay	5	60	35	Largely limestones and shales.	Subangular	32	A few hundred cu. yds.	

TABLE XII—*Con.*

Results of Physical Tests on Gravel and Boulder Clay Samples, Rocky Mountains Park

ALONG BANFF-WINDERMERE ROAD (Sinclair Pass—Kootenay Valley Division)

Sample No.	Type of material	Character of material						Amount available	Remarks
		Composition of pebbles				Shape of pebbles	% Clay and silt		
		Durable %	Intermediate %	Soft %	Remarks				
71	Gravel	0	70	30	Largely limestone and shales	Subangular	25	Very small.	
72	Boulder clay	0	90	10	" "	"	35	Fairly large.	
73	Clayey gravel	5	85	10	" "	"	25	"	
74	Gravel	5	70	25	" "	Rounded	33	"	Clayey loam.
75	"	5	65	30	Fresh blue limestone pebbles.	"	20	Large	Sufficient clay content for surfacing purposes.
76	"	5	85	10	" "	"	26	2,000 cu. yds.	Still better material than above.
77	"	10	75	15	" "	"	17	Large	Silt.

TABLE XIII

Results of Physical Tests on Gravel samples, Rocky Mountains Park. Sand mortar 1:3 mix

BANFF-LAKE LOUISE ROAD (Eldon-Laggan Division)

Sample No.	Fineness modulus	Per cent clay and silt	Colour test	Per cent water used	Per cent of strength relative to that of Standard Ottawa sand[1]				Remarks
					Tensile strength		Compressive strength		
					7 days	28 days	7 days	28 days	
7.....	2·56	7	Almost clear.....	13	128	114	161	129	Good concrete aggregate
8.....	3·24	4	100.....	12	96	92	155	144	" " "
11.....	2·82	6	Almost clear.....	12	125	117	171	139	" " "
13.....	2·70	9	Turbid.....	13	97	90	139	108	" " "
22.....	2·65	6	Clear.....	12	121	132	152	149	" " "
23.....	3·17	4	Almost clear.....	11	152	135	165	149	" " "
25.....	2·27	10	100.....	13	103	91	140	112	" " "
27.....	2·63	5	Almost clear.....	15	123	119	164	136	" " "
29.....	2·74	6	Turbid.....	14	115	114	106	108	" " "
30.....	3·07	2	Clear.....	12	131	118	115	121	" " "

ALONG BANFF-WINDERMERE ROAD (Vermilion River Division)

36.....	3·20	15	Almost clear.....	15	127	114	171	124	
37.....	0·80	33	Slightly coloured...	20	84	85	79	87	Too fine sand. Silty. Unsuitable.
45.....	2·43	21	" " ...	17	115	97	112	104	
51.....	3·32	8	Almost clear.....	13	143	126	163	105	

TABLE XIII—*Con.*

Results of Physical Tests on Gravel Samples, Rocky Mountains Park. Sand mortar 1:3 mix

ALONG BANFF-WINDERMERE ROAD (Sinclair Pass—Kootenay Valley Division)

Sample No.	Fineness modulus	Per cent clay and silt	Colour test	Per cent water used	Per cent of strength relative to that of Standard Ottawa sand[1]				Remarks
					Tensile strength		Compressive strength		
					7 days	28 days	7 days	28 days	
53.....	3·28	10	Slightly coloured...	14	149	154	197	160	Fairly high. Good concrete aggregate.
58.....	2·60	23	100.....	15	133	116	182	160	" " "
64.....	3·75	16	Clear.....	13	160	142	167	106	" " "
69.....	2·43	34	Turbid.....	22	72	66	62	57	Low on account of fineness of sand and clay coating on pebbles.
74.....	2·48	33	200.....	19	93	85	95	61	" " "
76.....	2·93	26	Turbid.....	18	93	80	111	85	" " "

[1]Ottawa Standard sand, strength of 1 : 3 mortar :

tensile, 7 days— 248 lbs. per sq. in.
" 28 days— 370 lbs. per sq. in.
compressive, 7 days—2,065 lbs. per cu. in.
" 28 days—3,525 lbs. per cu. in.

Amount of water used—11 per cent.

IV

EXPERIMENTAL ABRASION TEST ON CONCRETE

Samples of concrete for pavements were received on November 7, from Mr. E. Viens, chief of the Laboratory for Testing Materials, Department of Public Works, Ottawa.

The samples consisted of two sets of 15 concrete cylinders 2 inches in diameter and approximately 2 inches high, respectively made of limestone and trap rock aggregate graded up to $\frac{3}{4}$ inch.

The object of the test was to find out if the type of rock used as stone aggregate in the mixture would have any influence, and if so, to what extent, on the ability of the concrete to resist wear.

Description of the Test Performed

There is no standard test to measure the wear of concrete. However, wear tests are performed on concrete blocks in the Structural Materials Research Laboratories of the Lewis Institute at Chicago. The machine used is the Talbot-Jones rattler, which is built on the plan of a brick rattler.

The test performed on the submitted samples was carried on along the lines of the standard abrasion test for road stone, as an experiment. The modified Deval abrasion machine, with slotted cylinders, was used. In this modified machine all dust (material passing $\frac{1}{16}$ inch) is removed as fast as it forms; thus the possibility of a dust cushion, which in the case of soft material is responsible for erroneous results, is eliminated.

In the standard abrasion test for road stone a charge consists of about 50 pieces of stone weighing within 10 grams of 5 kilograms. The weight of material passing $\frac{1}{16}$ inch mesh that is worn off in 10,000 revolutions expressed as the percentage of the weight of the charge, constitutes the measure of the abrasion of the stone. It is known as the per cent of wear.

The samples handed us did not comply with the requirements of the standard test as to number and shape of pieces, weight, etc., consequently only comparative results could be expected.

Results of Abrasion Test

Per cent of wear at end of—	1,000	2,000	3,000	5,000	10,000 revolutions
Limestone agg...	6·27 p.c.	10·74 p.c.	14·30 p.c.	20·20 p.c.	32·75 p.c.
Trap aggregate...	5·80	8·62	11·04	15·20	23·00

The average per cent of wear obtained in the abrasion test for the two types of stone considered is as follows:—

	Limestone	Trap
Slotted cylinders method.....	8·25 p.c.	5·0 p.c.
(Closed " "	4·0 "	2·2 ")

In a general manner the tested concrete cylinders behaved like stone does in the usual test. A higher scale of figures was reached but the results are parallel with those generally obtained in the case of limestone and trap rock. In this test for wear the softer rock always shows larger increments in the per cent of wear as the number of revolutions of the cylinders increases. The figures given above indicate that in this respect the concrete cylinders have followed the rule. The differences in the per cent of wear at various stages of the test are in accordance with the usual results for limestone and trap rock.

It is important to note that at the conclusion of the test the concrete cylinders made of limestone aggregate were well rounded, with a perfectly smooth surface, while the trap rock concrete cylinders had kept more of their cylindrical shape and showed a rough surface.

It seems obvious from these results that the nature of the stone aggregate used in a concrete mixture has some influence on its resistance to wear, especially if the concrete is subjected to continuous and severe abrasive action. Moreover, if a slippery surface is to be avoided, a stone aggregate of good wearing qualities will likely yield a concrete pavement with a rough surface as it is worn under traffic.

TABLE XIV

Results of Physical Tests upon Bed-rock—Nova Scotia

Sample No.	Location	Type of rock	Specific gravity	Weight per cu. ft.	Water absorbed pounds per cu. ft.	Per cent wear	French coefficient of wear	Hardness	Toughness	Cementing value	Remarks
354	Nine-mile river, N.W. of Elmsdale.	Quartzite (Whin rock).	2·70	169	0·37	2·2	18·2	18·6	25	.....	Very durable road-stone.
355	S. of Oakfield sta., Truro rd.	" ..	2·68	167	0·75	2·9	13·8	18·6	12	.....	"
363	Hilden.....	Siliceous shale.....	.....	.....	.....	2·5	16	.....	.....	46	
356	2½ miles W. of Shubenacadie on Robertson rd.	Limestone.....	2·38	148	3·10	13·4	3·0	8·7	4	.....	Weathered and soft, unsuitable for road work.
365	2 miles W. of Shubenacadie on Robertson rd	"	2·49	155	3·38	4·7	8·5	.....	.....	.....	Rather soft material fresher than above.
364	S.E. of Kentville.....	Diabase.....	2·87	179	0·65	2·1	19	17	16	.....	Very durable.
357	N. of Centreville.....	Gabbro.....	2·90	182	1·18	3·1	13	17·1	20	.....	"
366	Along escarpment N. of Centreville.	Trap..... (Amygdaloid).	2·68	168	8·04	7·7	5·2	.....	.....	.....	Much weathered, unsuitable for automobile traffic.
358	Rock types entering into the composition of gravel samples collected.	Fine grained sandstone.	.....	.....	.....	3·2	12·5	.....	.....	low	
359	" " ..	Coarse grained sandstone.	.....	.....	.....	8·7	4·6	.....	.....	.....	Friable.
360	" " ..	Granitic rocks.....	.....	.....	.....	2·6	15·4	.....	.....	.....	Very durable aggregate.
361	" " ..	Quartzites.....	.....	.....	.....	2·4	16·7	.....	.....	.....	" "
362	" " ..	Trap (Amygdaloid)	.....	.....	.....	5·4	7·4	.....	.....	185	Soft, but cementing well.
	Ohio Road Quarry, 2 miles N. of Shelburne	Diabase.....	2·88	.....	0·2	2·4	16·7	18·5	14	.....	Very durable material
	Reid's quarry, Jordan Falls.	"	2·89	.....	0·5	2·5	16·0	17·8	16	.....	" "
	Black Point, below Liverpool.	"	3·12	.....	0·3	2·2	18·2	16·8	10	.....	" "
	Cherry cove, Lunenburg co.	"	2·87	.....	0·9	2·5	16·0	17·9	15	.....	Very durable material
	Sable river, Shelburne co.	Schistose, micaceous quartzite.	2·71	.....	0·4	2·1	19·0	16·7	14	.....	" "
	Sable river, Shelburne co.	Mica schist.....	2·78	.....	0·9	6·8	5·9	.....	.....	.....	Very soft.

TABLE XV

Results of Tests on Gravel Samples—Nova Scotia

Sample No.	Location of deposit	Granulometric analysis																Remarks
		Proportion of gravel to sand		Gravel							Sand							
				Per cent retained on screen							Per cent retained on sieves							
		Gravel per cent	Sand per cent	2½″	2″	1½″	1″	¾″	½″	¼″	8	14	28	48	100	200	% passing 200	
458	Near Waverly, along lake Thomas.	65	35	16	11	25	16	8	10	14	18	12	10	9	16	13	22	Hard quartzite gravel in clay matrix. Packs well on road.
486	Near Wellington.....	55	45	....	....	....	....	....	....	....	15	12	10	10	10	8	35	Contains clay. Used on roads with good results.
480	John Jordon's pit, 1 m. N. of Enfield.	50	50	....	5	16	29	10	13	27	20	13	14	21	20	8	4	Dark coloured, well packed gravel.
484	Nine-mile river, near Elmsdale.	70	30	26	13	14	16	9	9	13	23	17	24	28	6	1	1	
481	Dewis' siding, Shubenacadie.	30	60	16	18	9	9	6	16	26	27	20	20	15	10	4	4	Reddish brown, clayey.
478	Robertson's pit, 3 m. W. of Shubenacadie.	50	50	....	18	10	13	8	16	35	23	28	25	14	6	2	2	Dark coloured.
454	Upper Nine-mile river, pit at forks near cemetery.	15	85	....	....	....	48	7	13	32	5	8	40	38	5	2	2	Reddish brown iron oxide and organic matter.
457	N.W. of upper Nine-mile river, Ridge N. of road.	35	65	20	35	6	20	6	4	9	3	2	3	13	40	26	13	Sand, iron oxide stained.
466	Near Broadfield station....	35	65	....	16	22	24	11	10	17	10	20	45	20	3	1	1	
472	Robert Harvey's pit, Hilden.	65	35	17	12	7	19	11	24	10	38	19	19	12	6	2	4	Dark red coloured.
465	N.W. of Truro, along Amherst road.	80	20	28	12	12	13	10	11	14	39	19	15	11	8	4	4	Well cemented.
455	Along Amherst road W. of Masstown.	.....	100	....	....	....	....	....	....	....	....	....	6	42	25	20	7	Brick red sand, weathered. Little mica content. Too fine.
473	Gravel used on Amherst road near G enholme.	80	20	....	18	19	21	11	13	18	28	22	25	16	5	2	2	Reddish coloured. Well graded.
470	Mackenzie's pit, North river	75	25	25	10	15	24	7	7	12	34	20	20	15	6	3	2	Red coloured, well packed. Well graded.
453	North river.....	55	45	....	....	20	20	10	15	35	25	30	30	10	2	1	2	Well graded but lacks binder.
463	Alma, W. of New Glasgow.	75	25	....	11	22	32	13	12	10	18	24	29	12	4	4	9	Dark red coloured. Coated with $CaCO_3$

496	Near Telford, along Antigonish road.	70	30	6	2	16	21	13	16	26	31	22	19	12	7	3	6	Dark red coloured. Well graded.
467	Along Windsor-Chester rd. 1 m. S. of Windsor Forks.	45	55	....	10	25	20	10	15	20	20	22	26	18	8	3	3	Lacks binding power.
456	Church's pit No. 2, Falmouth.	.....	100	....	....	....	....	....	....	....	....	....	....	6	67	21	6	Dark reddish brown quartz sand Too fine.
479	Same as above.....	55	45	....	23	5	19	12	20	21	22	18	23	20	10	4	3	Well graded for road work.
485	Same as above.....	60	40	40	5	10	13	8	9	15	20	30	30	12	4	2	2	Pebbles coated with dark red clay.
468	Church's pit No. 1, Falmouth.	65	35	5	5	20	25	11	17	17	18	23	33	18	4	2	2	
459	Forks of Mt. Denison and Bog roads, W. of Falmouth	10	90	....	....	....	....	....	....	....	5	5	12	40	28	6	4	Sand with rounded quartz grains coated with red clay.
477	Ellis' pit, between Windsor and Hantsport.	15	85	....	....	15	15	10	10	50	25	28	24	13	6	2	2	Brownish coloured. Carries impurities. Good grading, but sandy gravel.
461	Along Bog road; near iron bridge, between Hantsport and Mt. Denison.	20	80	....	27	17	15	9	14	18	8	8	12	21	24	20	7	Pebbles clay coated. Reddish buff coloured. Organic matter Sand is fine.
460	Aberdeen beach, Mt. Denison.	.....	100	....	....	....	....	....	....	....	1	2	7	66	20	3	1	Yellow beach sand.
475	Oak Hill, Hantsport.....	60	40	....	15	8	15	12	20	30	20	13	23	30	8	4	2	Loam and organic matter.
471	W. of Wolfville, along main road.	60	40	6	18	12	13	9	13	29	35	35	20	6	2	1	1	Pebbles coated with $CaCO_3$. Although containing little fine, binds well on road.
483	W. of Wolfville.....	50	50	....	17	18	17	7	12	29	26	33	25	11	3	1	1	" "
464	Aldershot pit near cross road.	40	60	....	3	10	17	18	23	29	16	13	25	29	14	2	1	
474	S. of North Alton.....	50	50	....	10	11	16	13	20	30	25	18	17	16	9	6	9	Well graded, contains binder.
482	E. of Coldbrook.....	75	25	22	15	7	16	9	12	19	33	35	25	4	1	1	1	
462	Between Cambridge and Coldbrook.	45	55	....	30	19	16	10	10	15	17	18	27	24	7	3	4	
476	Centreville.....	10	90	....	....	....	17	18	17	48	10	20	40	20	6	2	2	Brick red gravelly soil.
498	N. of Centreville.....	.....	100	....	....	....	....	....	....	....	1	2	12	20	30	20	15	Weathered trap rock disintegrated into sand. Binds well.

TABLE XVI

Results of Tests on Gravel Samples—Nova Scotia

Sample No.	Character of Gravel							Tensile and Compressive Strength of 1 : 3 Mortars					
	Composition of pebbles				Shape of pebbles	Colour test	Fineness modulus	Water used %	Per cent of strength relative to that of Standard Ottawa sand [1]				Remarks
									Tensile strength		Compressive strength		
	Durable %	Intermediate %	Soft %	Remarks					7 days	28 days	7 days	28 days	
458	100	.....	.....	All qtz.....	Angular.....	Clear	2·02						
486	.....	.....	100	Black sl.....	Flat, ang.....	.....	1·83						
480	40	60	.....	D.—Gr. qtz. dark volcanic rocks..... I.—Ss.	Subang.....	Clear	2·56	15	49	91	83	132	Low at 7 days.
484	25	25	50	D.—Largely qtz..... I.—Gr. S.—Weath-ign-ss.	Rounded to subang.	Clear	3·17						
481	60	35	5	D.—Gr. qtz. trap..... I.—Lst, coarse gr.	Rounded.....	Clear	3·15						
478	50	15	35	D.—Qtz. gr. volcanic rocks. I.—Ss. S.—Weath. ss. sh.	Subang.....	600	3·36	13	121	135	192	216	High.
454	65	15	20	D.—Ign. rocks, gr. qtz.	Rounded.....	1,000	2·58	14	28	78	39	103	Very poor, due to fineness and impurities.
457	.....	25	75	Partly weath., reddish ss.	Subang.....	.....	1·80						
466	5	35	60	I.—Ss..... S.—Ss., sh.	Rounded.....	100	3·08	15	118	157	118	133	Very good.
472	15	60	25	D.—Qtz..... I.—Ss., ign. S.—Sh.	Rounded.....	800	3·53	14	94	122	150	184	Good.
465	50	35	15	D.—Gr., sy., dia., qtz. I.—Ss. S.—Soft trap.	Rounded.....	Clear	3·46	16	63	90	86	135	Low for grading, due to clay content.

455	.....	.....	.....	.....	.....	Clear	1·27							
473	90	10	.....	Mostly gr.....	Subang.....	400	3·40							
470	40	25	35	D.—Qtz..... I.—Ss. S.—Weath. trap and sh.	Subang.....	.....	3·46	13	86	93	121	147	Fair.	
453	5	5	90	80 per cent sh.....	Thin, flat.....	.....	3·57							
463	30	20	50	D.—Qtz..... I.—Ss. S.—Sh., weath. ss.	Rounded to subang.	.....	3·01							
496	.....	.....	100	Weath. volcanic rocks..	Angular.....	.....	3·31							
467	40	30	30	D.—Gr..... I.—Lst., ss. S.—Sch. weath. ign.	Rounded.....	600	3·10							
456	.....	.....	.....	.....	.....	Clear	0·79							
479	30	45	25	D.—Largely qtz..... I.—Ss., lst. S.—Sh.	Rounded.....	.....	3·01							
485	25	40	35	D.—Qtz..... I.—Ss., ign. S.—shaly lst.	Rounded.....	.....	3·38	12	136	145	228	236	Very high.	
468	20	65	15	D.—Qtz..... I.—Lst. ss. S.—Weath. trap.	Subang.....	.....	3·21							
459	.....	.....	.....	.....	Rounded.....	800	1·89							
477	.....	20	80	I.—Ss..... S.—Weath. sh., sl. ss.	Rounded and flat.	1,200	3·41							
461	.....	40	60	I.—Ss..... S.—Sh. sl. weath. trap.	Rounded to subangular.	1,500	1·74							
460	.....	.....	.....	.....	.....	Clear	1·86							
475	5	30	65	I.—Ss..... S.—Weath. black sh. and sl.	Ang. flat and rounded.	2,000	2·89							
471	25	30	45	D.—Qtz..... I.—Ss. S.—Sh., weath. volcanic	Rounded and flat.	Clean	3·89	11	127	144	211	213	Very high.	
483	10	60	30	D.—Ign. qtz..... I.—Ss, sl. S.—Weath. volcanic.	Rounded and flat.	500	3·62	11	137	151	207	216	Very high.	
464	25	25	50	D.—Qtz..... I.—Sch. S.—Sh. weath. trap.	Flat.....	.....	2·79	13	10	60	29	77	Unsuitable. Contains too much clay.	
474	.....	85	15	I.—Sch. sl..... S.—Weath. trap.	Angular..... and flat.	100	2·89	14	115	125	168	155	High.	
482	5	70	25	I.—Sl..... S.—Weath. trap.	Flat.....	Clean	3·89	13	79	93	119	134	Fair.	

TABLE XVI—*Con.*

Sample No.	Character of Gravel							Tensile and Compressive Strength of 1: 3 Mortars					
	Composition of pebbles				Shape of pebbles	Colour test	Fineness modulus	Water used	Per cent of strength relative to that of Standard Ottawa sand[1]				Remarks
									Tensile strength		Compressive strength		
	Durable	Intermediate	Soft	Remarks					7 days	28 days	7 days	28 days	
	%	%	%					%					
462	35	35	30	D.—Largely qtz. gr.... I.—Ss. S.—Soft ss. and weath. trap.	Rounded to subang.	500	2·93	12	72	88	114	110	Fair.
476	30	10	60	D.—Quartz, qtz..... I.—Ss. S.—Sh. ss.	.....	400	2·96						
498	.....	.....	.....	.....	.....	Clean	1·19						

[1]Ottawa standard sand, strength of 1 : 3 mortars:

tensile, 7 days— 261 lbs. per sq. in.
" 28 days— 292 lbs. per sq. in.
compressive, 7 days—1,550 lbs. per sq. in.
" 28 days—2,260 lbs. per sq. in.

Amount of water used—10 per cent.

Abreviations used in column for remarks:—dia., diabase; gr., granite; ign., igneous; lst., limestone; qtz., quartzite; sch., schist; sh., shale; sl., slate; ss., sandstone; sy., syenite; weath., weathered.

APPENDIX

PRELIMINARY REPORT

(On the investigation of Peat Fuel conducted by the Joint Peat Committee for the Federal Government and the Government of the Province of Ontario, from January 1, 1921, to March 31, 1922, together with a statement of plans of the work to be done during the year 1922.)

B. F. Haanel,
Secretary to the Committee.

Introductory.—The Peat Committee, shortly after their appointment early in 1918, carefully examined all existing reports and records concerning the manufacture of peat into a domestic and industrial fuel, and concluded that the air-dried machine peat process was the only economic one known to exist which gave promise of ultimate success when employed under conditions obtaining in Canada. To adapt this process to Canadian conditions the Committee found that new types of peat manufacturing machinery, capable of performing the different operations required in the course of manufacture, had to be developed. Consequently, the investigation inaugurated by them resolved itself into: first, design and construction; second, development; and third, operation of machines capable of operating under severe and adverse conditions with the employment of the minimum number of labourers and with the minimum charge for repairs and maintenance.

The preliminary reports issued by the Peat Committee from their appointment in 1918 up to and including the year 1920 dealt with the design, construction, development and operation of the two different types of peat machines—designated as plants Nos. 1 and 2—which the Committee decided offered the greatest promise of success. The investigation also included the design, construction and development of a small peat machine capable of being operated by one man and two boys, suitable for the manufacture of peat fuel—on a small scale—by individual farmers or groups of farmers, or small communities. This machine, during this period, also passed more or less successfully through the stages of design, construction and development.

As a result of the progress made up to the end of 1920, the Committee were in a position to make in their preliminary report for that year the statement: "Although it is not possible for the Peat Committee to state at this time that peat fuel can or cannot be manufactured in Canada on a commercial basis, the results of the investigation so far conducted enable them to arrive at the following conclusions:—

1. That the Anrep plant (No. 1), as it stands, is in no sense commercial.
2. That the Moore Plant (No. 2) under certain conditions can be employed commercially for the manufacture of peat fuel.
3. That the Anrep excavating element is the superior of the two and the logical one to employ, while—
4. The Moore spreading system is far more efficient and is the logical spreading system to employ."

Based on these conclusions recommendations were made for conducting the investigation through the season of 1921, and for concluding the same during the season of 1922-23. These were as follows:—

1. That a portable belt conveyer be constructed for operating in conjunction with the Anrep excavating element for the purpose of combining the best elements of the two types of machines.
2. To develop and commercially try out a small farmer's peat machine.
3. To operate the Moore plant throughout the entire working season on a strictly commercial basis, with the least number of men and without the supervision of the expert staff, in order to obtain valuable information and data concerning overhead costs, actual costs of laying fuel on the field, harvesting same and transporting to stock pile and loading into cars for shipment. Also to determine the loss in time through breakages, etc. It was also desirable that this machine be operated in order that sufficient peat fuel be manufactured to supply the market already created.

PREPARATIONS FOR 1921 OPERATIONS

This report is concerned chiefly with the recommendations just cited, and includes a somewhat detailed analysis of the results obtained with Plant No. 2, which was operated on a strictly commercial basis during the entire season in accordance with the above programme. This plant was not put into perfect condition inasmuch as this would have involved the redesigning and reconstructing of the caterpillar elements, an expense which the peat committee did not feel warranted in incurring, hence the results obtained are inferior to what would be expected were these known defects of the machine removed.

OPERATIONS DURING 1921

Improved Plant No. 4

On June 1 the first carload of the new conveyer parts arrived at Alfred station, and during the same month the remainder of the conveyer parts arrived at the bog. The assembling of the conveyer parts was immediately begun, and the installation of the entire plant was sufficiently far advanced on August 5 to enable the conveyer to be turned over under its own power, and on August 20 excavated peat was delivered to the belt of the portable conveyer. The time from August 20 to August 26 was spent in making adjustments in order to permit the belt conveyer, spreader and excavater to work in harmony, and on August 26 a very fair demonstration of the operations of the entire plant was made in the presence of the Peat Committee and others.

Portable Belt Conveyer.—The portable belt conveyer which takes the peat from the excavater and delivers it continuously to the spreader is of novel design and may even be described as a daring venture on the part of those responsible for the conception of the idea. Nothing of its kind had ever before been constructed, and although stationary belt conveyers of longer spans are in successful use, the attempt to employ a portable belt conveyer of the dimensions of this conveyer has never been made, so a great deal of credit is due to the engineer of the Peat Committee and to those responsible for the carrying out of the design and construction for

the remarkable success which was achieved almost on the very day when the conveyer was completely assembled. The conveyer is 850 feet long and is composed of ten 85 foot latticed boxed girder sections connected together, forming a flexible bridge member which is carried on eleven caterpillar elements, all of which are operated from the main caterpillar near the excavating element. This bridge supports a rubber-covered belt which is 850 feet between centres, thus making a total length of belt of 1,700 feet. Fears were entertained that the conveyer would get seriously out of line and throw the belt off when attempts were made to move it under its own power over the field, but these fears were entirely unfounded. In order to lay on the field ten tons an hour of standard peat fuel it is necessary to move the belt conveyer every $1\frac{1}{4}$ hour a distance of 13 feet 6 inches parallel to the line of travel of the excavating element, and this means a lateral movement of 108 feet in a working day of ten hours. The conveyer system, in fact, was put through a very severe test, a more severe test than it will ever be called upon to withstand in actual practice, and not only did it maintain its line, but, in general, its behaviour far exceeded the expectations of its designers. The spreader, which was designed and constructed for use with this conveyer system, was based upon the experience and results obtained with the spreader used on the Moore system. This spreader also had a very severe trial, and proved entirely satisfactory, with the exception of a few minor mechanical details which will be changed. The quality of the peat laid on the field by this improved plant is better than anything that has ever been obtained before at Alfred. Plate XIV shows the belt conveyer in place for operation. Plate XV shows the new spreader attached to the belt conveyer. Plate XVI is another view of the belt conveyer system and excavater, in the far distance, and shows the unloading device for delivering the peat from the belt into the hopper of the spreader. Plate XVII shows one of the caterpillar elements, of which there are eleven, and also the take-up pulley at the extreme outboard end of the conveyer. Plate XVIII shows the peat fuel which was laid on the field with this new machine. This plate shows very clearly the effect of the improved longitudinal and cross cutters. It will be seen that the peat blocks are separated by a considerable space, and that this increases the drying surface over that obtained with former methods for cutting. This improved method of longitudinal and cross cutting it is believed will greatly improve the quality and resisting properties of the peat, as well as materially reduce the time required for drying down to 25 per cent or 30 per cent moisture.

Plant No. 2—Moore Plant

This plant was constructed from original designs prepared by Mr. Moore, which were, before the formation of the Peat Committee, submitted to the Minister of Mines of the Federal Government who decided that it should be constructed. In designing this plant Mr. Moore's objective was the reduction to as great an extent as possible of the number of labourers required to operate the plant, and his design showed a great many novel features which it was believed would realize this end. Neither this plant, nor any plant of similar design, had previously been constructed, and it therefore devolved upon the government, and later the Peat Com-

mittee, to construct and develop it into a successful machine. The advantages which were claimed for this design at the time the design was presented were:—

1. Lower cost of plant for a given output as compared with the Anrep system.
2. Reduction in number of men required to operate, namely, seven men, compared with 15 or more for the Anrep system.
3. Four boys required to perform the operation of cubing, which required not less than 12 workers with the Anrep system.
4. Number of men required for harvesting reduced by one-half.
5. Fuel not loaded on railway cars automatically left in storage piles instead of remaining spread over the drying field.
6. No delays incurred in moving tracks or bringing machine back to starting point, which delays are responsible for the loss of 25 per cent to 30 per cent of the total working time, when the Anrep system was employed.
7. Extremely direct route of the peat from the working face to the drying field will permit using much less transmission machinery, and will provide more regular feed to the macerater and, therefore, increase its output.

The government, and later the Peat Committee, after carefully considering these claims were convinced that the design possessed sufficient merit to warrant experimenting with a full sized unit, and consequently the construction of this machine was authorized at the same time as the Anrep plant.

Erection of Plant on Bog.—Although the contract to construct this plant was given sufficiently early in the season to insure its erection on the bog in time to give the plant a complete mechanical test during the summer of 1918, failure on the part of the contractors to meet their obligations made it impossible to begin erection until late in the fall of 1918. Consequently it was not possible to investigate the mechanical behaviour of the plant until the spring and summer of 1919.

During this year the Moore plant had a severe mechanical trial, in the course of which a certain amount of peat was manufactured. This trial disclosed many weaknesses and faults in design, which were remedied to as great an extent as possible during the winter. The following year, 1920, the Moore plant was operated for the manufacture of peat fuel throughout the entire working season. That season's investigation demonstrated to the satisfaction of the Peat Committee that the original claims enumerated above were not borne out in every particular. For example, it was found that harvesting operations could not be performed with this machine without seriously reducing its capacity for making peat fuel. But a still more serious defect was disclosed, namely, the excessively long working face which this type of machine requires for laying down a season's production of peat fuel. In order to lay down one ton of peat fuel the Moore machine must travel a linear distance of 56 feet. This not only means that the entire weight of the machine, excavating element, conveyer and spreader must be moved that distance, but also that in performing a season's work it will be necessary for the machine to travel approximately

60 miles. This is a very severe handicap for any machine, inasmuch as the wear and tear on the entire plant to perform this excessive travel would be abnormal. Moreover, the fact that a very long working face, approximately two miles in length, is required to lay down a season's production of fuel, will limit its employment to a very few bogs. Even on bogs where such a machine could be employed it would be impossible to operate economically more than one unit to advantage. Another defect should be mentioned, namely, the difficulty of keeping the machine on the line of excavation. It has been observed that in order to keep the machine on this line, and insuring that the excavater buckets remove a constant quantity of peat, a very skilful operator is required, and even then the quantity of peat excavated in a unit of time may not be constant.

Novel feature of the Moore Machine.—The novel feature possessed by this machine is the method employed for transporting the excavated peat from the macerater to the spreader. This is accomplished by a belt conveyer which is flexibly attached to the middle of the main platform carrying the excavater and macerater. This belt conveyer is about 180 feet long, and is formed of a box girder about 30 inches square. The inside end of this girder is attached to the main platform, the balance of the weight being supported on a caterpillar element situated approximately 100 feet out, which is operated in unison with the caterpillars on the main machine through the medium of a power shaft connected to the caterpillar driving mechanism on the main machine. The spreader is attached to this belt conveyer at any desired position and is dragged along by the main machine. When one row is completed the direction of the main peat machine is reversed and the spreader is removed to the opposite side of the girder and shifted the required distance along the conveyer. After this row is completed the main machine is again reversed and the operation repeated until the requisite number of rows representing the season's output are laid on the field.

Plate XIX shows the belt conveyer with the spreader in position near the end farthest removed from the line of excavation, and the entire plant at the end of the working face ready to be reversed. This plate also shows clearly the peat laid on the ground. Plate XX shows a close up view of the spreader in the above position, and clearly illustrates the operation of the cutting devices. It also shows the position of the spreader when the maximum number of rows are laid. The elevator at the extreme end of the girder is used to convey the dried peat to stack or small tram cars.

When the last row is completed the machine must either stop or else the first row laid down must be sufficiently dry for harvesting in order that the spreader may be placed in its original position and the operation of laying down peat again proceeded with as described. Plate XX shows also the device for unloading the peat from the conveyer belt to the hopper situated at the centre of the spreader. This hopper delivers into a box the full width of the spreader (12 feet) which is provided with a 14 inch standard spiral conveyer, one-half of which is right hand and the other left hand, both sections tapering from the full size at the centre to practically nothing at the ends. This spiral is rotated by power received from the transmission shaft, which extends the full length of the bridge work

and insures that the peat fed in the hopper will be spread in a sheet of uniform thickness. The depth of the opening for the discharge of the peat on the field may be adjusted from 3 inches to 6 inches, and this opening extends over the full length of the box. By means of this regulation the thickness of the sheet of peat laid on the bog can be varied to suit conditions.

Drying field required for season's output.—For an average production of 60 tons per day of ten hours, this machine must travel 3,360 lineal feet, and since a maximum of about 40 days is required for complete drying, allowing for Sundays, it is necessary to provide a drying field which will permit the laying out of approximately 120,000 feet of spreader section, that is 120,000 feet by 12 feet (the width of the spreader) times the depth of the peat spread. The belt conveyer is of sufficient length to provide for the laying down of 12 rows. The drying field for ten hours per day operation should, therefore, be 10,000 feet long in order to insure the complete drying of an average production of 60 tons of fuel per day. This is one of the chief drawbacks of the Moore plant. The working face provided for this machine at Alfred was 8,000 feet long, but this did not permit of the complete drying of the fuel and at the same time the production of the quantity of peat fuel for which the machine was designed.

Harvesting.—When the machine was developed on paper it was considered possible to employ the above conveyer for harvesting the dried peat at the same time that the peat slop was being laid on the field, but this operation it was found could not be efficiently performed without further experimentation. In short, all attempts to employ the conveyer for harvesting the dried peat fuel resulted in a decrease in the quantity of peat produced. Other methods of harvesting were consequently resorted to.

Difficulties encountered and improvements made.—The most serious difficulties encountered in connection with the operation of the Moore plant were occasioned by the caterpillar elements. When these were designed the engineer had very little data to guide him, since no machine capable of performing the operations required of a peat machine had ever before been designed for travelling over a spongy surface on caterpillar elements, consequently, serious defects were observed almost immediately in the design of the caterpillars and the mechanism driving them, and also in the method employed for steering the machine. Consequently, most of the time and money which has been spent on alterations and improvements to this machine were expended on the caterpillar elements. As a matter of fact the caterpillar difficulties have not yet been eliminated. They could easily be improved, but it was not considered necessary to rebuild the caterpillars for the purpose of completing the investigation of this machine, since it was decided, at the close of 1921, to cease operating the Moore plant.

Minor difficulties were also encountered in the excavating element, and in other portions of the plant, but these were or could be eliminated without much trouble.

CONCLUSIONS ARRIVED AT AT THE END OF THE SEASON OF 1920

At the conclusion of the field operations at the beginning of the fall of 1920 a large amount of data was available for comparing the relative merits of the Moore and Anrep plants.

1. It was clearly demonstrated that the Moore plant showed a marked economy in labour over the Anrep plant, but that in spite of this economy the Moore plant could not compete with the Anrep plant, provided the Anrep plant were equipped with an improved delivery and spreading system.
2. It also appeared from the observations made during the operation of the Moore plant that the maximum length of the bridge work which can be supported by a single caterpillar had been practically reached and that any extension of this bridge work would make the machine altogether too cumbersome. Twelve rows of peat, therefore, appears to be the maximum number of rows which can be spread with a machine of the Moore type. As stated before, a linear travel of 56 feet is required in order to lay down one ton of standard peat fuel, and a working face 10,000 feet long for a ten hour shift and 20,000 feet long for two ten hour shifts. If it were possible to lengthen the belt conveyer bridge work to accommodate double the number of rows of peat laid, that is to say, the laying of 24 rows instead of 12, the length of the working face would be cut in two. This would remove certain of the difficulties under which the plant labours, but, as stated, it does not appear that the length of the belt conveyer can be sufficiently increased without making the entire plant too cumbersome and unwieldy.
3. The advantage which it was also expected the Moore plant would have over the Anrep plant in harvesting did not materialize. This was due to two causes: (*a*) When harvesting operations were being carried on as originally intended a stoppage anywhere along the line from the excavater to dumping into storage meant a stoppage of the entire equipment. (*b*) Variations in the weather could not be provided for, that is, an exceptionally dry spell would permit the drying of the fuel to proceed too far before the fuel could be harvested, and if the weather were abnormally wet there was the further danger of the whole plant being stopped or held up waiting for the fuel on the ground to dry. On account of these two difficulties it was decided that it would be more efficient to harvest the production of the Moore plant with the harvesting equipment employed with the Anrep plant—even though this equipment was not designed for operation in connection with the former—than to attempt to harvest with the machine itself as was originally intended.
4. The quality of the fuel manufactured during the early part of the season compared unfavourably with that produced by that of the Anrep plant. This was in part due to the inferior quality of peat in that portion of the bog where the Moore plant operated, but it was also partially due to the fact that the long working face over which the machine takes only one complete cut before a complete row is laid down, is likely to freeze to a considerable depth in a cold winter, and that the first rows which are made from the peat excavated from a frozen face would produce peat of an inferior quality. In the case of the Anrep plant the excavation is made over a very short working

face, consequently, it is possible to dig away the frozen face before the operation of making peat is begun. A certain quantity of peat manufactured with the Moore plant during the season of 1920 was of inferior quality due to this cause.

5. In order to carry out in the most economical manner the various operations in the manufacture of peat fuel, according to the air-dried machine-peat process, it is necessary that the plant and machinery employed be as simple as possible, since it is lnot feasible to have skilled workmen on hand to operate a p ant which may be situated at some not easily accessible point, and which, moreover, can operate for only a part of the year. Any peat plant, therefore, which requires for its operation skilled labour suffers a severe handicap. The Moore plant to a certain extent is under this handicap, since to operate it to the best advantage more skilful and more careful operation is required than with the Anrep plant. Moreover, the uniformity and quality of the fuel laid down by the former depends to a large degree on the care exercised by the operator. This applies to a certain extent also to the Anrep plant, but its operator has a much better control over his machine and does not therefore require to exercise the same degree of skill.

On account of the above defects, which were observed in the operation of the Moore plant and the defects which were noticed in the spreading system employed with the Anrep plant, the Peat Committee decided to recommend that the best features of both plants be incorporated into a combination plant. The following table gives the principal data obtained during the operation of the Moore plant during 1919, 1920, and 1921.

TABLE I

—	1919	1920	1921
Total length of working season exclusive of Sundays		105 days	107 days
Total days actual operation		(a) 41 days	62·3 days
Unavoidable lost time		18·4 days	12·4 days
Percentage of season		17·5 p.c.	16·6 p.c.
Avoidable lost time		45·6 days	32·3 days
Percentage of season lost through delays which might have been eliminated		43·4 p.c.	30·1 p.c.
Distance travelled, about	45,000 ft.	149,250 ft.	242,250 ft.
Fuel laid down	(b) 500–600 tons	(c) 2,665 tons	(d) 3,889·5 tons
Fuel laid down per hour average		3·4 tons	5·2 tons
Total labour cost to operate plant	$1,023.82	(e) $4,603.33	$3,380.27
Total labour cost for repairs		$1,138.50	$700.77
Labour cost to lay out, per ton		(f) 1 75	0·869
Cost of cubing per ton		(g)	0 20
Total labour cost ready to harvest			1·07
Total labour cost if troubles eliminated			0·807
Maximum day production		53·6	70 tons

(a) Does not include 180 hours night work which is included to arrive at average production per hour in 1920.

(b) This distance travelled should have produced 800 tons but no fuel was made during the first three or four cuts when working face was being opened up.

(*c*) It is estimated that 2,985 tons of fuel were made by No. 2 in 1920 on a basis of 50 lineal feet of row laid out per ton, and harvesting records would seem to confirm this figure, but 1921 records show it requires 56 lineal feet of row to produce one ton of 30 per cent moisture fuel, and this figure has been used so as to make a fair comparison with 1921. The apparent greater production in 1920 is accounted for by the fact that the fuel harvested was not dried down to 30 per cent moisture content.

About 1,600 tons of fuel made by No. 2 in 1920 sold for $4 per ton.

About 200 tons of fuel made by No. 2 in 1920 was used for fuel.

About 865 tons of fuel made by No. 2 in 1920 was not dry enough to harvest.

(*d*) Estimates of fuel made in 1921 on basis of 56 lineal feet of movement of spreader check very closely with actual weights.

Total fuel weighed into storage	3,194·5	tons
Total fuel used for fuel	221·0	"
Total fuel wasted on field in harvesting	116·0	"
Total fuel left on field not dry	358·0	"
	3,889·5	tons
Known loss in production through this spreading, about	450·0	"
Total	4,330·5	tons
Theoretical production for 242,250 feet at 56 feet per ton	4,326·0	

(*e*) This figure includes $735 cost of night shift and $700 paid to two additional men over regular shifts who were necessary to look after suction hose through which water was drawn to the boiler. For a fair comparison with 1921 this latter figure should be subtracted from the total, making cost per ton.

(*f*) ..... $1.47.

(*g*) Cost of turning and cubing in 1920 for the first part of the season was done by day labour. A great deal of work was done on old frozen peat in an attempt to save it, and the total cost for turning and cubing in 1920 is abnormally high and any figure given would be misleading. In the latter part of the season cubing was done by contract for 20 cents per ton.

TABLE II

Summary Trouble Sheet Plant No. 2, 1921

Cause	Unavoidable		To be expected		Due to known defects which could have been eliminated	
	hrs.	mins.	hrs.	mins.	hrs.	mins.
1. Weather	26	45				
2. Turning	..	..	..	..	11	55
3. Cross ditches	..	..	..	..	3	15
4. Oiling	..	..	..	..	7	20
5. Move from field to field	..	..	..	..	20	40
6. Engine	..	..	4	0	4	35
7. Excavater element	..	..	10	0	14	45
8. Roots in excavater	..	..	1	45	..	..
9. Chains	..	..	8	0	52	17
10. Belt	..	..	1	10	..	..
11. Spreader	..	..	12	0	39	35
12. Cross cutter	..	..	4	5	..	..
13. Ties	..	..	..	10	..	..
14. Sprockets	..	..	8	20	..	..
15. Clutches	..	..	4	0	8	5
16. Gears	..	..	1	0	..	..
17. Late starting	..	..	..	..	0	50
18. No steam	..	..	..	..	10	30
19. No water	..	..	..	..	14	33
20. No fuel	..	..	..	..	..	40
21. Roots in macerater	..	..	15	0	22	15
22. Iron in macerater	..	..	10	0	31	45
23. Variable speed	..	..	13	..	13	10
24. Loader	..	..	..	..	25	0
25. Stacker	..	..	..	..	0	45
26. Harvesting	..	..	..	..	3	40
27. Repairs	..	..	18	20	36	45
	26	45	97	50	322	20
	2·675	days	9·78	days	32·23	days

During 1919 this plant was given a mechanical test and only a small quantity of fuel was manufactured in the course of experimentation. In 1920 an effort was made to operate the plant during a full working season. The total length of the working season, exclusive of Sundays, was 105 days, and the total days during which the machine was actually operated was 41. It will be noted that the total days of actual operation during 1921 were greatly increased and that the unavoidable lost time was reduced from 18·4 days in 1920 to 12·4 days in 1921, which corresponds to 17·5 per cent and 11·6 per cent respectively of the entire working season. The avoidable lost time during 1921 was notably decreased under that of 1920, namely, from 30·1 per cent to 43·4 per cent. A very large increase will be noted in the lineal distance travelled by the plant in 1921 as compared with that travelled in 1920, and the average fuel laid down per hour in 1921 is notably higher than that laid down in 1920, namely, 5·2 tons, which is about 8/10ths of a ton from the maximum which was expected. It would appear, therefore, from the results of these three seasons that this plant could be brought to a high degree of efficiency, and that most of the mechanical troubles experienced could be entirely eliminated, which would make it possible to lay on the field the requisite quantity of fuel during an entire working season, provided a 10,000 foot working face was available. It is necessary at this point to direct attention to the fact that no attempt was made to put the machine in perfect condition for operation during the season of 1921, since to do so would involve a large expenditure of money which the Committee could not recommend. This machine, therefore, began operations under a severe handicap, which was in no sense decreased as the season advanced.

Regarding the total labour costs to operate the plant in 1921 as compared with 1920, it will be noted that for a greater production of fuel the labour cost is one-third less and that the cost for repairs is also in the vicinity of one-third less than in 1920, but the decrease in labour cost is more noticeable in the costs calculated per ton of fuel laid down, namely, in 1921 $·869 while in 1920 it was $1·75. This shows a reduction of approximately 50 per cent. If the troubles above mentioned were eliminated the labour costs for laying down a ton of fuel would be reduced to practically 81c. In 1921 the maximum daily production realized was 70 tons, while that of 1920 was 53·6 tons. The total quantity of fuel manufactured during 1921 was 3,889 net tons of 2,000 lbs., 450 tons were lost through spreading; the total production was consequently 4,439 tons. This checks very closely with the theoretical quantity which should have been produced for a lineal travel of 242,250 feet, namely, 4,326 tons. Table No. II gives a summary of the troubles experienced with the machine during the season of 1921, the time lost and the reasons therefor.

Costs

While this investigation is being conducted for the purpose of developing machinery capable of performing all the operations required to convert the raw peat substance of a bog into a marketable fuel with the greatest degree of efficiency, such efforts would result in nothing but waste in time and money were the cost of the fuel so produced too great to permit competition with other classes of available fuel now on the market.

The main objective has, therefore, been to develop machinery which can not only perform the various mechanical operations in the most efficient manner possible, but also produce a fuel at an economic cost. This has not proven to be a simple problem. The removal of one difficulty which barred the road to success often introduced another equally or more serious, and, as often happens in the development and refining of processes and their application commercially, the Peat Committee were confronted with the fact that the refining of mechanical appliances can be carried to a point where the saving resulting from the higher efficiency of labour-saving devices is more than offset by the increased cost. Consequently, the problem with which this investigation is chiefly concerned, resolved itself into the co-ordination of manual labour and mechanical labour-saving devices so that the human element and the mechanical element would be properly balanced.

In order to produce a low grade fuel like peat from a very low grade substance such as raw peat, which contains about 90 per cent water and only 10 per cent combustible matter and ash by weight, with machinery capable of performing the various operations, to a large degree, automatically, the output of finished fuel per unit of time must be as large as possible. This implies a large and powerful manufacturing unit which in itself introduces a serious difficulty, viz.: overhead costs.

Plant No. 2 was complete with labour-saving devices which partially defeated the very object for their introduction, and its elimination as a machine for manufacturing peat fuel on a commercial scale generally was partly due to their defects. A detailed estimate of the total cost of one ton of standard peat fuel as manufactured by this machine as it stands would consequently be of little or no value. In the light of the experience gained during two complete seasons operations, a new machine of the Moore type could be built at a much lower cost than was possible with the original machine, and, moreover, the new machine would have incorporated into its design improvements which would correct many of the defects of the former, and would as a result be far more efficient and under special conditions even prove a commercial success.

With the results and records of operation now at the disposal of the Committee it is possible to roughly estimate the cost of producing standard peat fuel with plant No. 2—redesigned and rebuilt. However, it must be understood that these costs are subject to revision in the final report, when detailed estimates of the costs of a complete plant will be obtained.

The estimated cost of making fuel with the Peat Committee Plant No. 2 is based on these conditions:—

1. Plant rebuilt in light of all knowledge gained.
2. Plant is electric motor driven.
3. 10,000 feet working face available.
4. Minimum wages 35c. per hour.
5. 100 days minimum working season.
6. 10 hour shift.

And that the capacity of the machine when redesigned and rebuilt will reach an average of 60 tons of standard peat fuel per day for 100 days. There is no reason to doubt that this, and even a greater production would under these circumstances be realized. For convenience the details of the estimates are placed under the following headings:—

A. Capacity of machine.

B. Cost if operated by electric motors—power supplied from stationary steam generating plant.

C. Costs if plant is driven by self-contained steam plant.

D. Costs if plant is driven by semi-Deisel engine.

The subject matter under these heads is sufficiently clear to require no further explanation.

A. Capacity of Machine:—

Plant No. 2 moved 242,250 feet in 1921 in 62·3 days elapsed time, of which 4 days were spent in shifting to new position.

Therefore Plant No. 2 moved 242,250 feet in 1921 in 58·3 days actual travel, or 415·5 feet per hour. But observed speed of travel of machine was over 8 feet per minute, or 480 feet per hour.

Therefore Plant No. 2 lost 7·75 days during 1921 operation not recorded in trouble sheet. Of this the greatest part was due to holding back when using its own harvester, and from travel records it is estimated this amounted to 4–5 days and the balance, say 3·75 days, was lost due to short delays which were unrecorded.

There is no reason why an improved No. 2 plant should not sustain a rate of travel of 8 feet per minute when in motion, equal to 480 feet per hour, equivalent to production of 8·5 tons of fuel per hour.

Known unavoidable delays	=11·6 p.c.	
Unrecorded delays	= 3·6 "	
Delays due to shifting machine to new position on the field (on 10,000 ft. face)	1·8 "	
Total delays	17 "	
Therefore		
Capacity of plant should be 83 p.c. of 8·6 tons		7·0 tons per hour
or per 100 days		7,000 tons
Deduct for fuel	400 tons	
left on ground 3 p.c	210 "	
waste at loading 5 p.c	350 "	
Say		1,000 tons
Net amount available on cars		6,000 "

B. Cost if operated by electric motors—power being supplied from stationary steam generating plant.

1. Production Cost: Items which vary as the number of tons manufactured:—

Cost of raw materials on royalty basis per ton		0·05
Operation—		
Runner per day	5·00	
Man in excavater	3·50	
Spreader	3·50	
Spare man	5·00	
Electrician and mechanic	7·50	
Fireman	4·00	
Water boy and messenger	1·50	
Total per day	30·00	

Allowing 25 days to open up and close down for winter, 125 days at $30 per day—Total labour cost $3,750, or per ton		0·625
Supplies, oil, etc		0·015
Turning by contract		0·15
Harvesting to storage or railway cars		0·60
Total production cost		1·44

2. Overhead:—

Estimated capital investment—		
No. 2 machine complete with motor drive	$10,000 00	
Harvester for same	1,200 00	
16 lb. track 12,000 ft	5,000 00	
12 lb. track 2,000 ft	600 00	
Harvester cars 12	1,500 00	
Locomotive	1,500 00	
Storage and loading equipment	3,000 00	
Railway siding	1,500 00	
Office, machine shop, power house and transmission	15,000 00	
Total capital layout	$39,300 00	
Add for under estimate and emergencies 10 p.c	3,930 00	
Total, say	$45,000 00	
Total production cost		1·44
For capital charge write off for all purposes—		
20 p.c. of $45,000.00	$ 9,000 00	
or per ton for 10 hour shift		1·50
for 16 hour shift	0 95	
for 20 hour shift	0·75	
For executive overhead charge—		
Manager for year	$2,400 00	
Clerk 8 mos. at $75	600 00	
Office expenses	600 00	
Total	3,600 00	
Or per ton on 10 hr. shift		0·60
16 hr. shift	0 37½	
20 hr. shift	0 30	
Add to re-handle from storage to cars		0·35
Total cost for 10 hr. shift		3·89
16 hr. shift		3·11
20 hr. shift		2·84

C. Similar Costs, if plant driven by self-contained steam plant:—

Raw material		0·05
Production cost—		
Fireman	4 00	
Engineer	7 50	
Runner	4 00	
Man in excavater	3 50	
Man to get fuel	3 50	
Man on spreader	3 50	
Spare man	5 00	
Mechanic helper	3 50	
Water boy	1 50	
Total	36 00	

Or for 125 days=$4,500.00 per ton		0·75
Add supplies		0·015
Turning		0·15
For mechanic if manager and mechanic not same man		0·10
Harvesting		0·60
Total production cost		1·665
Overhead—		
No. 2 machine complete with steam power	$15,000 00	
Harvester 1	1,200 00	
Track 16 lb	5,000 00	
Track 12 lb	600 00	
Cars	1,500 00	
Locomotive	1,500 00	
Storage and loading equipment	3,000 00	
Siding	1,500 00	
Office and machine shop	2,500 00	
Total	31,800 00	
Plus 10 p.c.	3,180 00	
Say	35,000 00	
Or for 10 hours—overhead charges per ton		1·17
Executive overhead charge same		0·60
Re-handle charge same		0·35
Total		3 79

D. Similar Cost, if driven by semi-Deisel engine:—

Raw material		0·05
Production cost—		
Engineer	$7 50	
Runner	4 00	
Man in excavater	3 50	
Man on spreader	3 50	
Spare man	5 00	
Mechanic	6 00	
Mechanic helper	3 50	
Water boy	1 50	
	$34 50	
Or for 125 days—$4,312.50 per ton for 6,400		0·68
Supplies		0·025
Turning		0·15
Harvesting		0·60
Total production		1·505
Overhead—		
Total capital investment not greater than for steam		
Or annual charge of $7,000 for 6,400 tons or per ton		1·10
Total executive charge same as for steam or annual charge of $3,600 for 6,400 tons or per ton		0·56
Re-handling charge		0·35
Oil fuel 1 gal per H.P. at 11c.—per ton		0·11
Total		3·62½
Summary—		
Separate steam driven power plant		3·89
Self contained steam driven power plant		3·79
Self contained crude oil engine		3·62½

FARMER'S PLANT

Plant No. 3 is in design a miniature Moore-Anrep combination plant. The method of supporting the excavating and macerating element is based on the principle employed with the Anrep system, while the delivery and spreader system employs the principles so successfully demonstrated

in the Moore plant; but in order to combine the features of these two plants on a small and economical scale new problems arose which had to be solved. In the course of its development many discouragements were experienced, and it appeared at one time as if the entire design would have to be changed; but an alteration to the excavating macerating element, and a rebalancing of the other parts of the machine, together with the installation of a more powerful motor eradicated many of the worst difficulties, and a few further changes put the machine into such condition that the Committee see no reason why it cannot be successfully operated during the coming season.

Like the larger machines, this small peat plant must also provide means to excavate, macerate and spread the pulped peat on the ground. In order to carry out these operations in the most simple and economical manner an effort was made to design an element which would combine, in one, the operations of excavating and macerating.

As the machine now stands it is expected that its average hourly capacity will be between 1 and 2 tons of standard peat fuel. This will amount to 10 to 20 tons during a working day of 10 hours. The machine is very simple to operate and can be easily handled by one man and two boys.

MACERATING

The degree to which maceration can be economically conducted has an exceedingly important bearing on the quality of the fuel produced. It is therefore essential that every effort be made in attempting the manufacture of peat fuel according to the wet process to obtain the highest degree of maceration consistent with economy. The results which have been obtained during the past three years have led the Peat Committee to conclude that the Anrep macerater as at present constructed is not entirely satisfactory for operation where a high rate of production is required, since it has been their experience that a large amount of the time which has been lost during a limited season of 100 days during which it is possible to manufacture peat fuel, has been due to stoppages of the plant through choking or breaking of the macerater with roots and pieces of iron, which now and again find their way into it. To avoid loss of time through such stoppages, the Committee made arrangements with the Jeffrey Manufacturing Company, of Montreal, to experiment with a swing hammer shredder which they were led to believe might prove successful for macerating peat. This shredder arrived at the plant too late for experimentation, but a few minutes demonstration run, which it was possible to carry out, showed that the maceration obtained with this machine was far superior to that possible with the Anrep macerater. This shredder will be tried early in the spring of 1922, and if it is found unsatisfactory, two Anrep maceraters placed in parallel will be employed. Arrangements for installing them are now under way. This will make it possible to obtain the maximum production of peat fuel during an entire working day, and will avoid stoppages, since it is not probable that both maceraters will be choked up at the same time. Moreover, while one is being cleaned the other can be overloaded for a short time, thus insuring the delivery of a more or less continuous supply of peat to the belt conveyer.

LOSS OF PEAT FUEL THROUGH FIRE

On November 8 fire was discovered in the stock pile of fuel which contained a portion of this season's production, a quantity of fuel manufactured last season, and several hundred tons of broken frozen peat and fines. The fines were stored in this pile preparatory for shipment as a fertilizer filler and soil conditioner. The portion of the stock pile where steam and smoke were seen to issue was immediately opened up and efforts made to quench the incandescent fuel thus exposed. Towards the evening of this day it appeared to those in charge that the fire had been put out, or at least got under control, since smoke ceased to issue from any part of the pile. But a rising wind soon revived the fire, which broke out in other parts of the pile, and though strenuous efforts were made to locate the seat of the fire, this proved impossible, since it spread with great rapidity when it reached the good peat fuel, where an abundant supply of air was available. The meagre and inadequate equipment available which could be used for fighting a fire proved of little avail, and efforts to cut the pile in two, in an attempt to separate the burning peat from the good peat fuel, failed owing to a change in the direction of the wind, which blew the poisonous gases resulting from incomplete combustion on to the workers and necessitated their hasty withdrawal. Consequently it was possible to salvage only a very small quantity of fuel.

Prior to November 8 the superintendent of operations and the engineer of the Peat Committee, who examined the stock pile on many occasions, had no reason whatever to suspect that the peat was on fire, and moreover, the great rapidity with which it developed appears to indicate that it was of very recent origin.

A thorough investigation failed to reveal the cause. It may have been due to carelessness on the part of a labourer, or to a spark from a locomotive, which, lodging in the fines, was covered by freshly dumped cars and slowly ignited the fines at a considerable depth. But the theory that the fire was the result of spontaneous combustion can be given little credence since there is no record of a peat fire originating in this way in European countries, where it is the practice to store great quantities of peat in this manner.

Hereafter the peat will be stored in separate piles. This will permit a fire to be attacked more efficiently—should a fire ever occur again—and at the same time insure the minimum loss, inasmuch as a fire in one pile would be confined to that pile.

However unfortunate such a loss may be, no blame, in light of the evidence obtained, can be attached to any member of the staff responsible for the conducting the investigation at Alfred.

The stock pile before burning was over 22 feet high, from 30 feet to 40 feet in width at the top, and about 100 feet in width at the bottom. These dimensions will give some idea of the difficulties encountered in the attempt which was made to cut the pile in two.

It is estimated that the loss by fire was:—

	Quality	Quantity	Estimated value
1921	Good peat fuel	1,780 tons	$8,500
1920	Poor disintegrated fuel	600 "	1,800
	Screenings	100 "	300
	Damage to trestle	.....	400
			$11,000

SALE OF FUEL MANUFACTURED

The Peat Committee has met with considerable difficulty in disposing to the best advantage of the fuel manufactured. This has been occasioned partly by the nature of the work which the Peat Committee is conducting, namely, an investigation which is concerned more with the development of the machinery for manufacturing peat fuel than with the product. Consequently, the quality of the fuel placed on the market has not in certain instances been of the best. In order to dispose of the fuel made during a season in the most economical manner it has been found impossible to avoid shipping a certain portion of inferior fuel with the good fuel, and a small quantity of the former has therefore found its way into households and has given rise to a certain amount of dissatisfaction. While it is necessary to call attention to this fact it is gratifying to know that the complaints of inferior fuel have been exceedingly few, and that by far the major portion of the fuel sold since the inauguration of the Peat Committee's investigation, so far as the Committee could ascertain, has given complete satisfaction.

The operation of two experimental machines with the consequent laying out of fuel on two sections of a bog widely separated, made it impracticable to install harvesting and loading apparatus which would enable these operations to be carried out in the most economical manner. The installation of the proper facilities for efficiently harvesting and handling the fuel manufactured would have involved a comparatively large capital expenditure, which the Peat Committee did not feel warranted in incurring, on account of the temporary character of the work they are conducting. Harvesting operations will be discussed under a separate heading.

The policy of the Peat Committee is to dispose of the fuel manufactured to householders, in order to introduce this fuel for domestic purposes. A considerable portion, however, was sold for heating office buildings, and to a certain extent the government buildings of Ottawa, but care was taken not to permit the fuel, in wholesale lots, to get into private hands.

The selling of the fuel in Ottawa and Ottawa district was put in the hands of a selling agent, who undertook to dispose of the entire output. The agreement which the Peat Committee made with him called for the production of 4,000 tons, but this agreement was broken by both the sales agent and the Peat Committee; first, by the former in not accepting delivery of peat as required, and later in the season by the Committee owing to the fire which destroyed a large portion of the season's output. The selling and distribution of peat fuel presents certain problems which must be worked out independently. They are, however, all practical problems which can be handled more satisfactorily by business men. Peat sales for 1921 were as follows:—

Raw peat sold in cars	1,160	tons at	$5.00	$5,800 00
Locally	125	"	5.00	625 00
Old peat sold in cars	18	"	2.50	
	85	"	3.00	300 00
Screenings sold in cars	113	"	3.00	339 00
Peat on hand saved from fire and sold for future delivery, estimated	50	"	3.75	187 50
Approximate total sales				$ 7,251 50
Less contract in connection with sales in Ottawa and Hull, estimated			$460 00	
Joliette Steel Company			248 75	
Car screenings to fuel testing station			71 16	
			$759 91	759 91
				$6,491 59

HARVESTING

The harvesting of the finished peat fuel, which comprises the operations of taking the fuel off the ground, loading onto small tram cars, transporting these to loading trestle, and unloading small cars onto storage pile or into freight cars for shipment, has presented many problems, certain of which must be worked out independently. Prior to the war these operations have been largely performed by hand labour, but during and since the war labour conditions have changed very materially, as regards wages, and in order to carry out these operations in the most efficient and economical manner, manual labour must be replaced to the largest extent by mechanical devices. When the Peat Committee undertook this investigation they were of the opinion that the harvesting of a season's production of fuel would not present any problems of an unusual nature and that the ordinary methods could be employed without increasing unduly the cost of the fuel. As the investigation proceeded, however, it has been discovered that harvesting constitutes one of the most important problems, second only to the manufacture of the fuel itself, and that if this operation is not conducted in the most economical manner possible, the economies realized from the development of peat fuel manufacturing machinery will be more than offset by the cost of the former.

As time permitted close attention was given to the problems presented in harvesting, and automatic devices were designed, constructed and tried for performing certain operations heretofore performed by manual labour. While considerable progress has been made in this direction, the problem is by no means entirely solved and economies in advance of those so far obtained may be expected. Harvesting a season's production of the No. 4 plant—the combined Moore-Anrep plants—can be performed at a less cost than was the case with plant No. 2, but with the latter plant little attention was paid to this particular problem, when it was realized that harvesting could not be performed economically by the machine itself. If plant No. 2 had proved satisfactory for commercial operations, generally, instead of for special isolated cases, attention would have been paid to the problem of harvesting a season's production of fuel laid down with that machine.

The peat investigation was undertaken at a time when the cost of raw materials entering into the construction of peat machines and general equipment required for the peat manufacturing plant were inordinately

high. At this time also labour wages reached the highest level. To-day these conditions have materially changed. Peat manufacturing machinery and other equipment required can be purchased for practically 50 per cent of war prices, and labour wages have also dropped materially. The figures of cost which have been prepared for this report are consequently based on higher labour wages than it is expected will obtain during the season of 1922-23.

METEOROLOGICAL OBSERVATIONS WITH REFERENCE TO DRYING

Meteorological observations, including precipitation, temperature, and barometric pressure, have been made during the season of 1920 and 1921. A full record of these observations, as well as a complete report on the drying of peat fuel as observed at Alfred will be included in the final complete report. It will suffice to say here that the observations covering the last two seasons have shown that the maximum period during which peat fuel can be manufactured—that is, laid on the ground and dried sufficiently for shipment—is 100 working days.

It has been ascertained that peat manufacturing operations can begin as early as May 1, and continued up to August 25.

OUTLINE OF INVESTIGATION 1922-1923

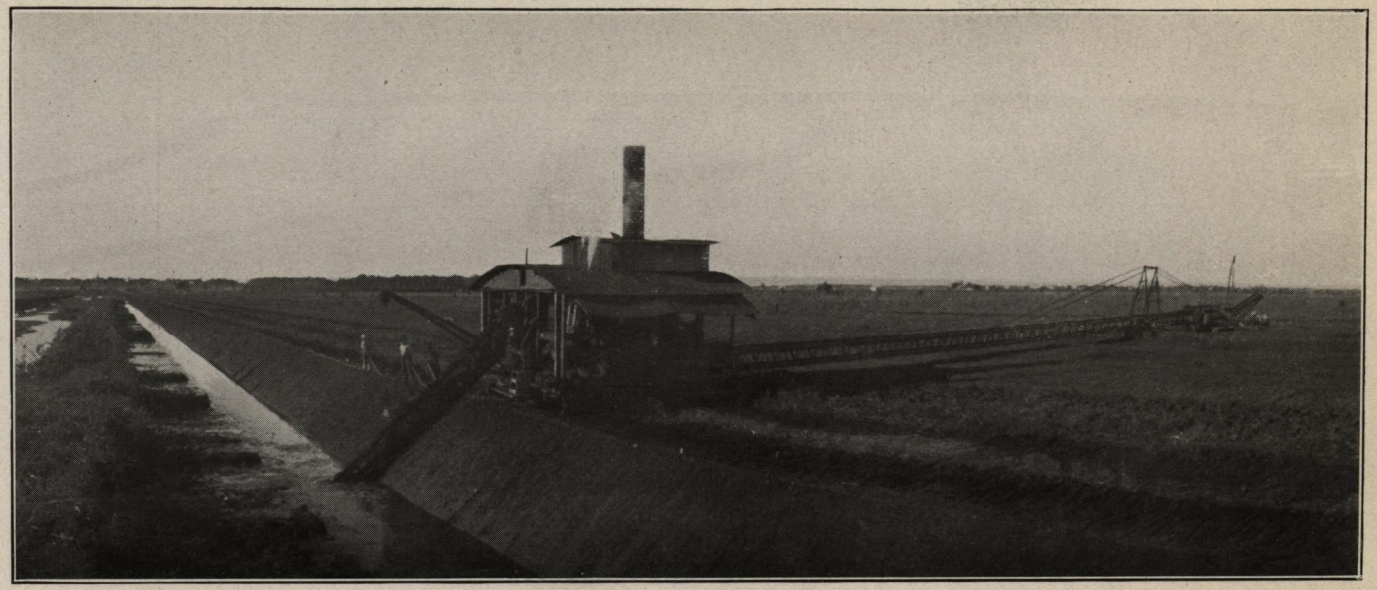
In accordance with the programme submitted in the spring of 1921, the investigation for 1922 will be confined to the operation of the new combination plant and the small plant No. 3 on a commercial scale, the former through an entire working season, and the latter for a sufficient period to demonstrate its value for the work it was designed to perform. The Jeffrey swing hammer shredder, which was shipped to the bog last fall too late for experimentation will be thoroughly tested early this spring, and if the results are satisfactory it will be employed during the working season for macerating the peat excavated by the combination plant. It is not expected that any new equipment will be purchased, nor will any construction be undertaken with the exception of that involved in the replacing of the loading and storage facilities, which were destroyed by fire. Additional machinery which may be required to carry to completion the investigation will, if possible, be rented instead of purchased. Arrangements have already been made to rent at a nominal cost an engine for operating the Hammermill shredder in case it is satisfactory, or the two Anrep maceraters in parallel. This engine, if purchased, would have cost the Committee considerably over $2,000, while on the rental basis it will involve an expenditure of approximately $400 to $500.

Just as soon as the snow is off the ground the new combination plant will be put into condition for beginning commercial operation about May 1. Plant No. 2 will be dismantled, with the exception of the boiler and propelling engine, which it is proposed to employ for supplying the extra power required to operate the new plant.

The harvesting equipment has already been overhauled so that this is in order for the coming season's work. In order to render the combination plant commercial in every respect an hourly average production of ten tons of saleable peat fuel must be maintained throughout a working season of 100 days. There is no reason to believe, however, that this

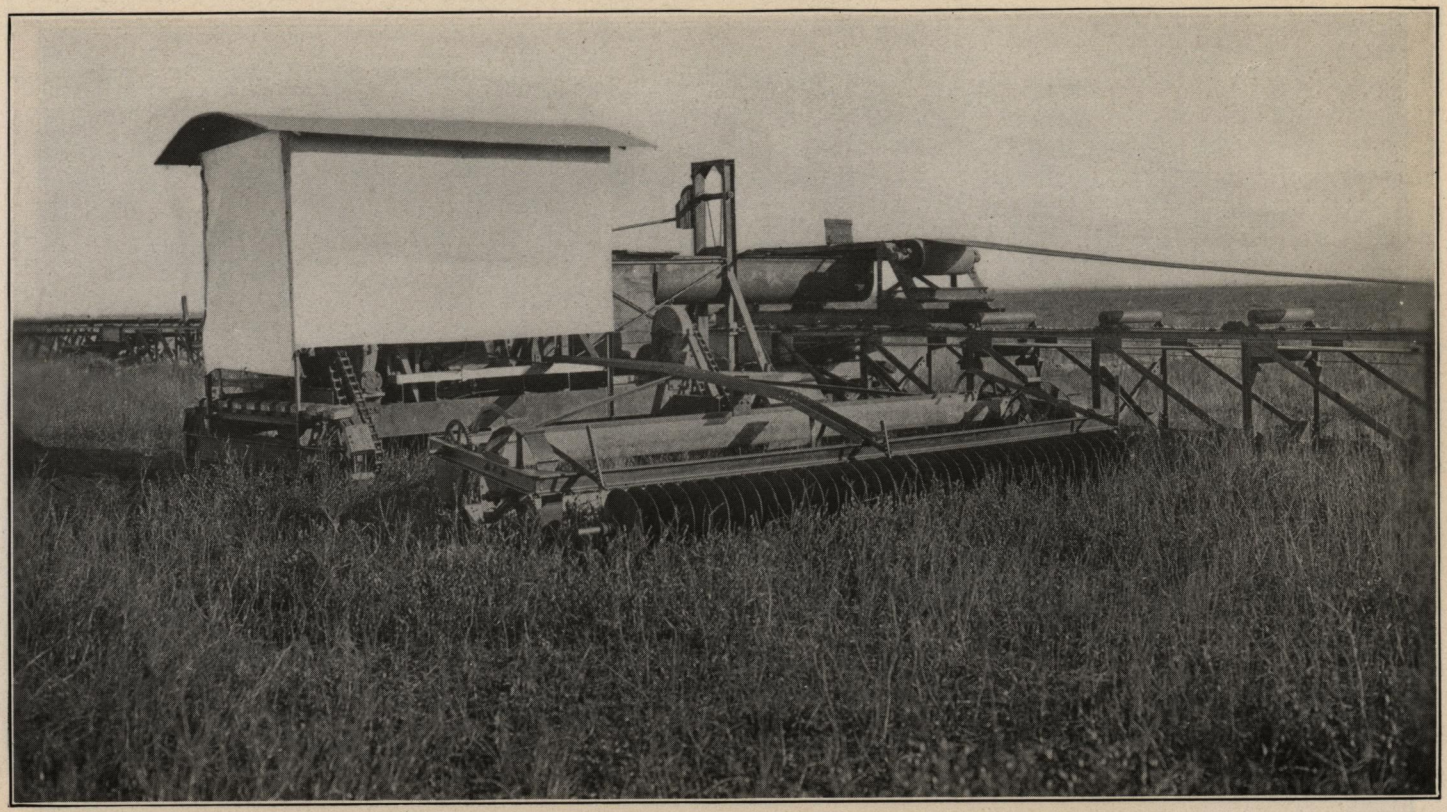
production will not be realized. An hourly production of ten tons of saleable peat fuel for ten hours a day and for a working season of 100 days means laying on the field a greater quantity than 10,000 tons of finished peat fuel. To accomplish this a longer working face will be required than was necessary for the Anrep plant as it was originally operated, and to provide the additional working face it is planned to operate on both sides of the main excavation according to the accompanying plan. This shows two drying fields, Nos. 1 and 2. It also shows the semi-permanent tracks which will be laid for harvesting purposes, and the permanent tracks at both ends of the field for shifting the belt conveyer from No. 1 drying field to No. 2.

At the conclusion of the working season in 1922 the preparation of the final report of the operations of the Peat Committee since the inauguration of the investigation in 1918 will be immediately begun.



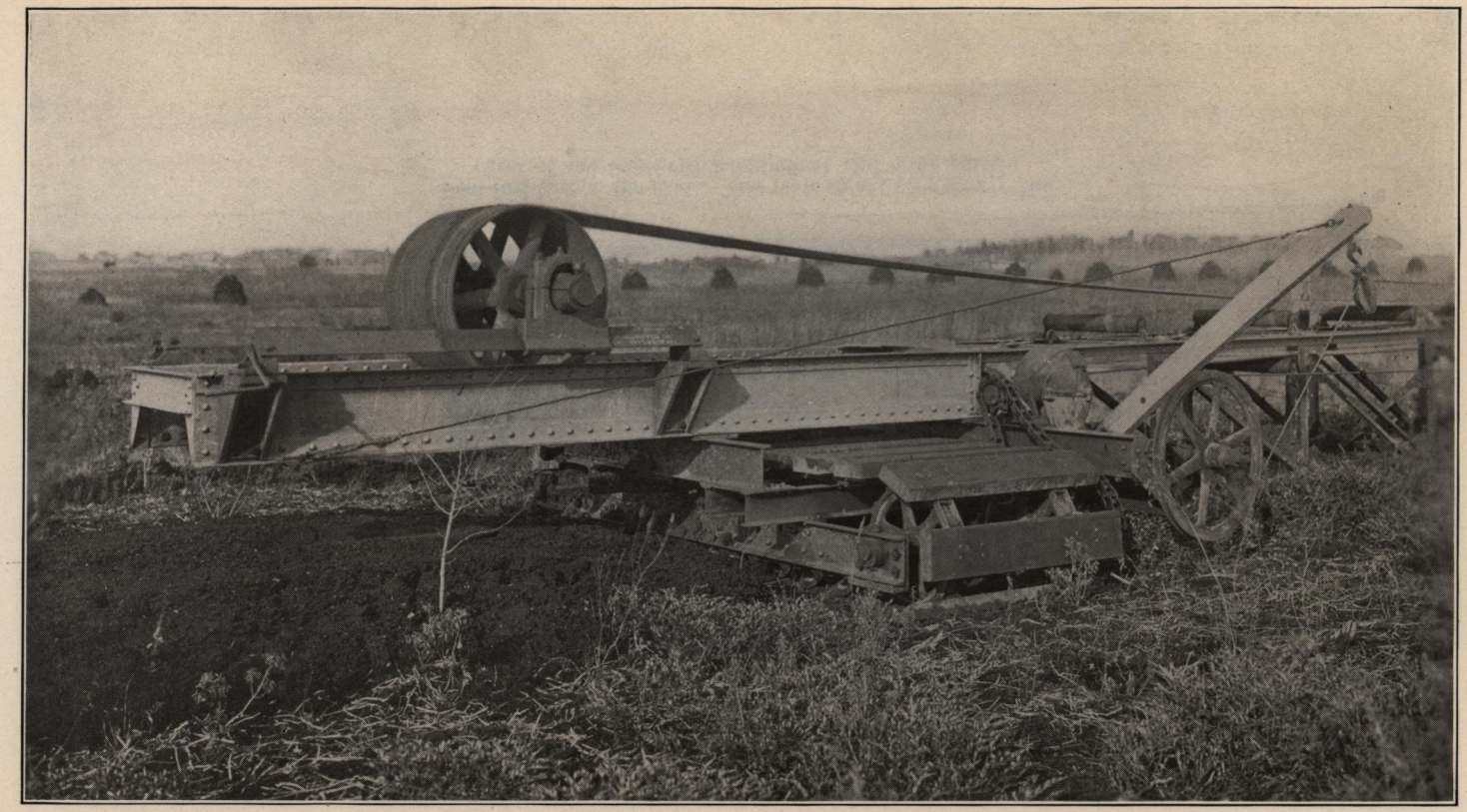
Plant No. 2 in operation

Belt conveyer in place for operation



New spreader attached to the belt conveyer

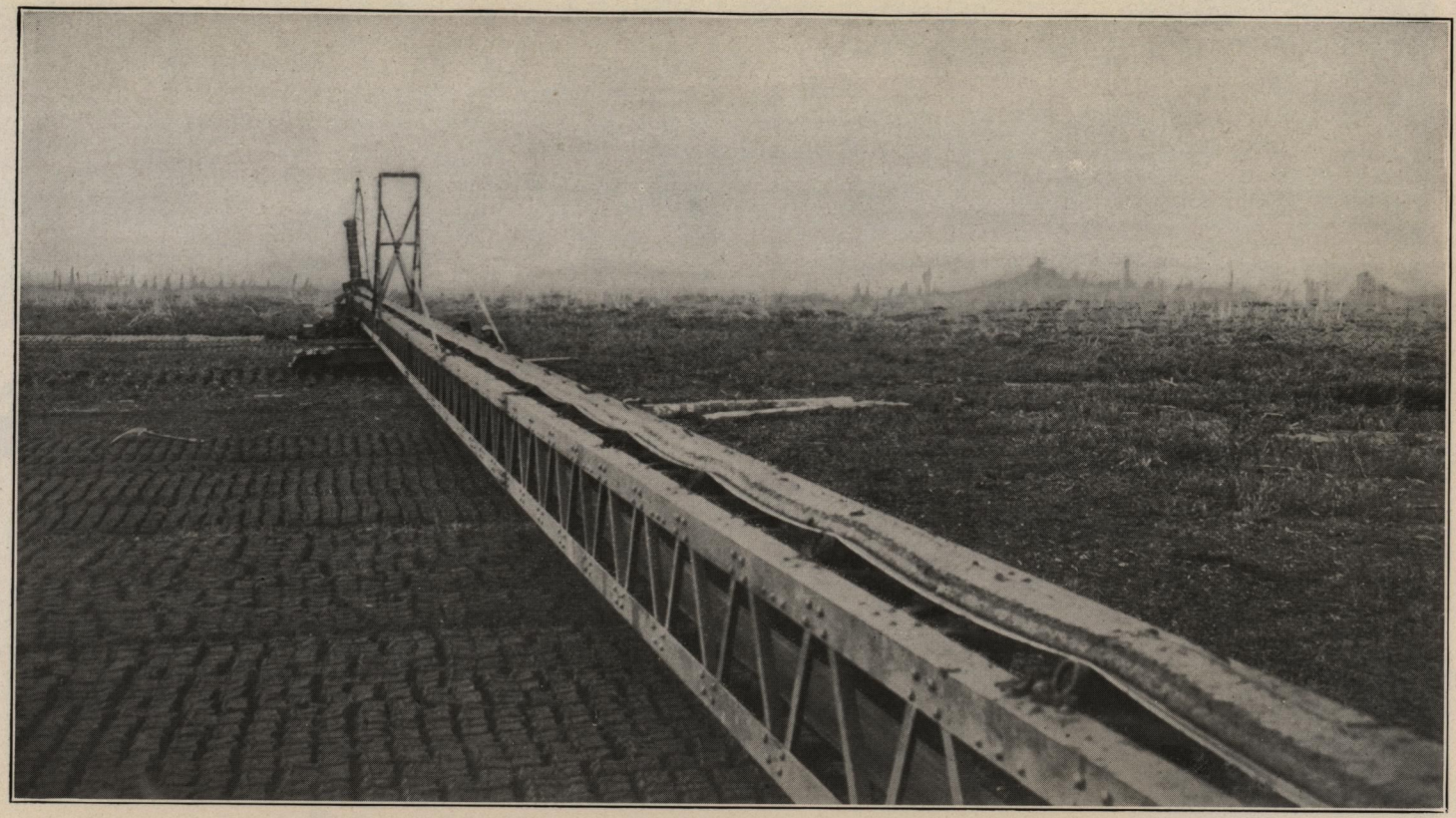
Belt conveyer system and excavater showing unloading device for delivering the peat from the belt into the hopper of the spreader



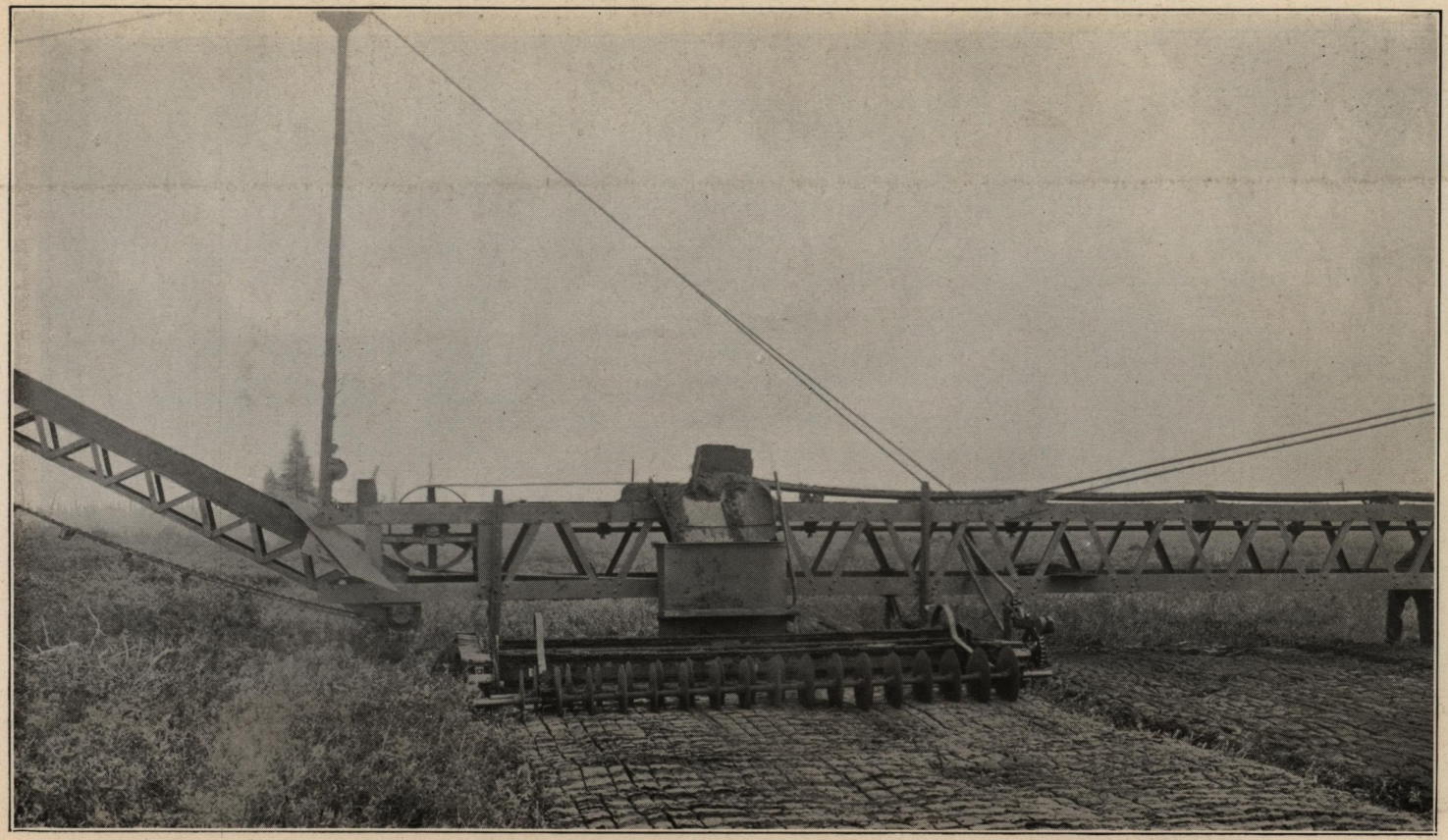
One of the caterpillar elements, and take up pulley at extreme outboard end of the conveyer

PLATE XVIII

Peat fuel laid on the field. This plate shows very clearly the effect of the improved longitudinal and cross cutters



Belt conveyer with spreader in position near the end farthest removed from the line of excavation, and the entire plant at the end of the working face ready to be reversed



Close up view of spreader showing operation of the cutting devices; also position of the spreader when the maximum number of rows are laid. The elevator at the extreme end of the girder is used to convey the dried peat to stack or small tramcars.

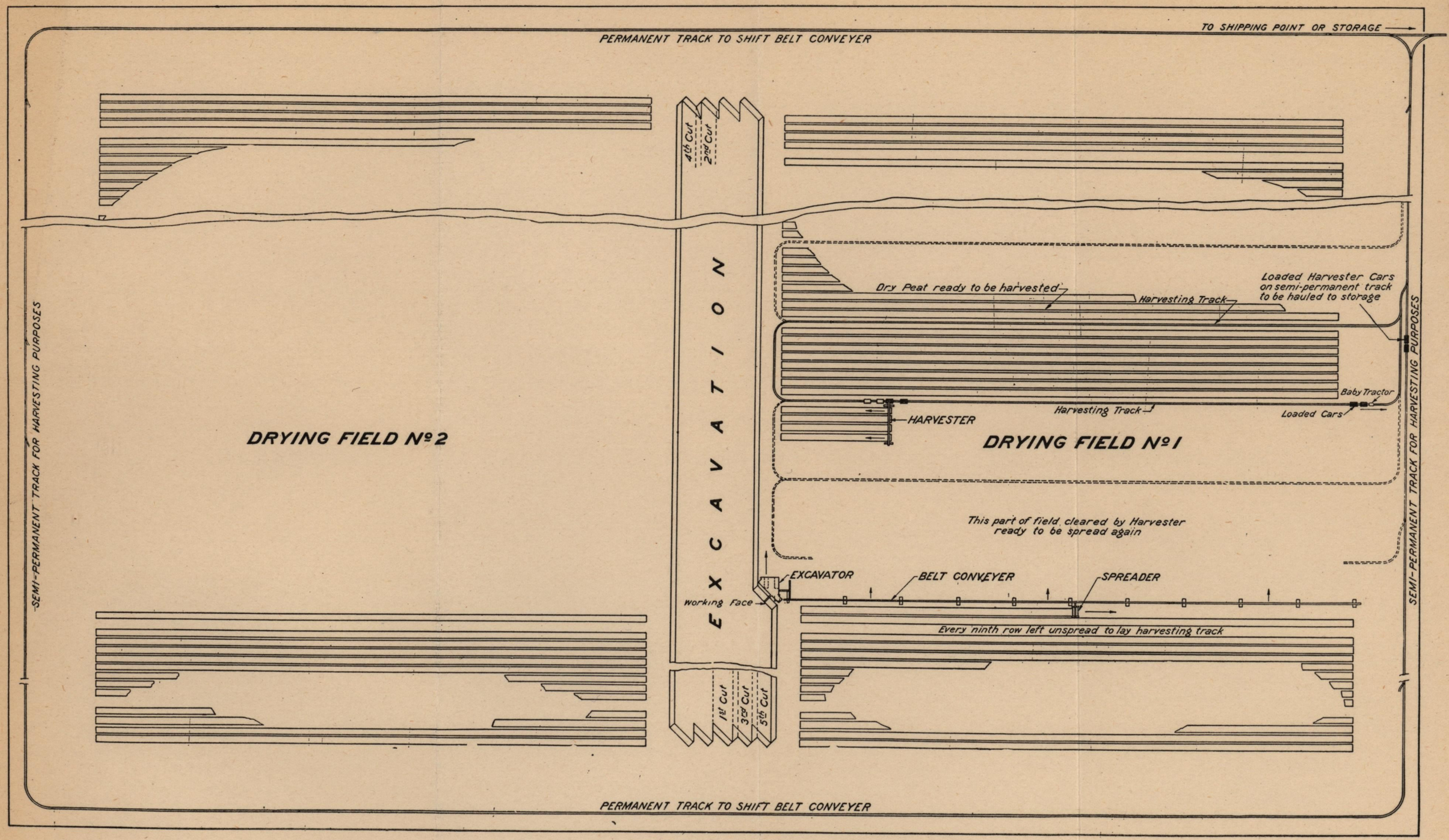
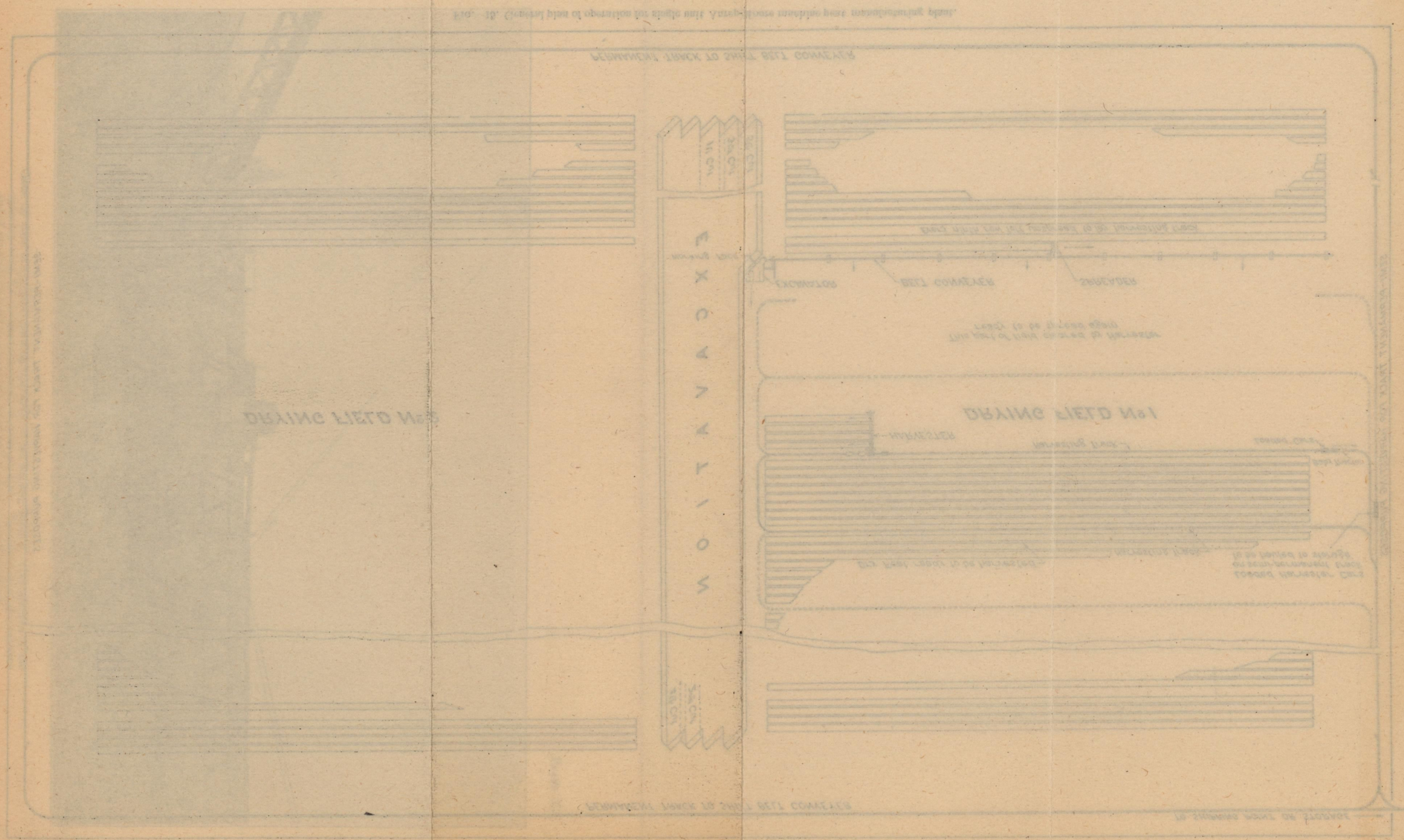


FIG. 15. General plan of operation for single unit Anrep-Moore machine peat manufacturing plant.



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